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PROCEEDINGS

OF THE

AUSTRALASIAN
Institute of Mining Engineers.

VOLUME III.—No. II.

JUNE, 1907.

EDITED BY THE SECRETARY.

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PUBLISHED BY THE INSTITUTE,
AT THE HEAD OFFICE, MELBOURNE.
1907.

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Australasian Institute of Mining Engineers.

LIST OF OFFICERS, 1907.

President:

Frank A. Moss, Kalgoorlie, W.A.

Vice-Presidents:

R. W. Chapman, Adelaide, S.A. | C. F. Courtney, Melbourne, Vic.

Councillors:

A. H. Merrin	Melbourne, Vic.
F. Danvers Power	Sydney, N.S.W.
Robert C. Sticht	Queenstown, Tas.
H. Herman	Melbourne, Vic.
John Warren	Nymbool, Q.
D. B. Waters	Dunedin, N.Z.
G. D. Delprat	Broken Hill, N.S.W.
R. Hamilton	Kalgoorlie, W.A.
H. Lipson Hancock	Moonta, S.A.
Stanley Hunter	Melbourne, Vic.
A. S. Kenyon	Melbourne, Vic.
A. Montgomery	Perth, W.A.
G. A. Richard	Mt. Morgan, Q.
H. H. Schlapp	Melbourne, Vic.
C. A. Süssmilch	Sydney, N.S.W.

Hon. Auditor:

Thos. F. Brennan, Melbourne, Vic.

Secretary:

D. L. Stirling, Melbourne, Vic.

Local Correspondents:

Western Australia	E. H. B. Macartney, Kalgoorlie.
South Australia	J. C. Fraser, Adelaide.
New South Wales	F. Danvers Power, Sydney.
Queensland	William Poole, Charters Towers.
Tasmania	Leslie Jolly, Launceston.
New Zealand	D. B. Waters, Dunedin, N.Z.

The Executive Committee consists of all Members of the Council residing, or, for the time being, in Melbourne.

Head Office:

57-59 Swanston Street, Melbourne, Victoria.

Vol. III. JUNE, 1907. No. II.

INSTITUTE MATTERS.

The Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

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MINUTES OF MEETINGS.

ANNUAL MEETING.

MELBOURNE, 31st January, 1907,

In the Institute Rooms, Colonial Mutual Chambers,

Market-street, at 12.30 p.m.

Mr. E. A. Weinberg occupied the chair.

The notice convening the meeting was read, and the Chairman declared the meeting properly constituted.

The minutes of the First Ordinary Meeting, 1906, were read and confirmed.

The Report of the Council for 1906, and Financial Statement were read and adopted.

The result of the ballot for the election of officers, as reported by the scrutineers, was as follows:—

Officers and Council.

President	Frank A. Moss, Kalgoorlie, W.A.
Vice-President	C. F. Courtney, Melbourne, Vic.
Council	G. D. Delprat, Broken Hill, N.S.W.
	R. Hamilton, Kalgoorlie, W.A.
	H. Lipson Hancock, Moonta, S.A.
	Stanley Hunter, Melbourne, Vic.
	A. S. Kenyon, Melbourne, Vic.
	A. Montgomery. Perth, W.A.
	G. A. Richard, Mt. Morgan, Q.
	H. H. Schlapp, Melbourne, Vic.
	C. A. Süssmilch, Sydney, N.S.W.

REPORT OF THE COUNCIL FOR 1906.

To the Members.

The following outline of the work done by the Institute for the year ending 31st December, 1906, is submitted.

The number of Members on the Register is now 275 classified as follows:—2 Honorary Members, 2 Fellows, 1 Life Member, 252 Members, 13 Associate Members, and 5 Students. Twenty-six Members and 7 Associate Members were elected during the year. Five Members were removed from the Register.

The Council regrets to have to report the loss by death of two members, *viz.*:—Mr. W. T. Batchelor, of Queenstown, Tasmania; and Mr. E. V. Bensusan, of the Gold Coast Colony, West Africa.

Two meetings of members were held during the year—the Annual Meeting, 1906, at Melbourne on the 12th January and the First Ordinary Meeting at Adelaide in May.

Papers received from members during the twelve months were published in Volume II., No. IV., and Volume III., No. I., of Proceedings. Volume XI. of Transactions was issued in July.

The Executive Committee held eleven meetings during the year.

The First Ordinary Meeting 1906, the business part of which was held in Adelaide, S.A., included excursions to Wallaroo, Moonta, Port Pirie, Iron Knob, and Broken Hill. The minutes of this meeting and an account of the excursion appeared in Volume III., Number I., of the Proceedings. At the invitation of the Council no less than sixty-five students from the mining schools and a number of members of kindred societies accompanied the members of the Institute on this excursion. The meeting was a most important event in the history of the Institute, and the Council desires to place on record its cordial thanks to all who helped to make it such a gratifying success.

Preliminary arrangements are being made with a view to holding the First Ordinary Meeting, 1907, at Kalgoorlie during the month of July or August.

The alterations to the Rules, proposed by the Council, and submitted to members in May, were unanimously approved. The scope of the Institute has thus been considerably widened. It now provides for all those connected with the science and practice of Mining or Metallurgy. Special meetings are not now required for the examination of ballots for the election of members—the duty devolving on the Council. The entrance fees previously charged to members have been abolished. The number of members on the Council has been increased from twelve to eighteen. The positions of Secretary and Treasurer as elective and executive officers have been done away with, the Council now having the filling of these posts.

In reponse to an invitation, Mr. H. M. Ridge was appointed to represent the Institute at the Third International Colliery Exibition in London during the month of June, 1906.

During the year a five-years lease of rooms in a new building, erected in Swanston Street, Melbourne, was signed on behalf of the Institute in conjunction with the Royal Victorian Institute of Architects, the Victorian Institute of Surveyors, the Victorian Institute of Engineers and the Chamber of Mines of Victoria. These rooms are centrally situated and should be a great convenience to members. It is anticipated that they will be ready for occupation by the end of February. Some delay has been caused in the erection of the building through a strike in the building trade.

Separate prizes of ten guineas each were offered to students and past students of the mining educational institutions of Australasia for the best paper on any subject connected with the mining, geology, treatment, reduction &c. of Australasian Ores. The result of the Competition is not to hand.

The prizes offered to Students who took part in the excursion to Broken Hill in May were awarded as follows:—The First Prize of Ten Guineas (The gift of Mr. Aaron T. Danks) F. W. Sewell, Bairnsdale School of Mines. The Second Prize of Five Guineas (The gift of Mr. Donald Clark) A. D. Galloway, Ballarat School of Mines, The Special Prize of Two Guineas for a Geological Essay (the gift of Mr. A. E. Kitson) H. C. Richards, Melbourne University. Thirteen Students entered for the competition.

Mr. Wm. Poole, Director of the Charters Towers School of Mines has been appointed to the vacancy caused by the resignation of Mr. G. A. Richard as local correspondent for Queensland.

The Mining and Geological Institute of India, the Bureau of Geology and Mines, Rolla, Mo., and the West Australian Mining Building and Engineering Journal were placed on the exchange list during the year. The Institutions on that list now number 140. Additions to the library amounted to 180 volumes, making a total of 1563 volumes.

The Financial Statement for the year ending 31st December, 1906, is attached and shows a credit balance to current account of £27 9s. 2d. and of £400 on fixed deposit. The amount received for members subscriptions shows a fair increase over the previons year.

In conclusion the Council desires to remind members that every facility is now afforded for the publication and dissemination of information concerning the mining industry. The importance of collecting and recording data and the great mutual benefit derived by compilation and discussion and the investigation of engineering and metallurgical problems will, it is hoped, stimulate members to even greater efforts in contributing papers or sending criticism and additional information on those written by others.

On behalf of the Council,

G. D. DELPRAT, *President.*

STANLEY HUNTER, *Treasurer.*

A. S. KENYON, *Secretary.*

MELBOURNE, 31*st* *January*, 1907.

AUSTRALASIAN INSTITUTE OF MINING ENGINEERS.

Balance Sheet, 31st December, 1906.

LIABILITIES.				ASSETS.			
Publications—Proceedings, Vol III., No. I.	£41	18	0	Outstanding Subscriptions	£119	14	0
Sundry Printing	2	0	9	Advertisements	22	0	0
Prizes	10	0	0	Office Furniture	10	0	0
Meeting Expenses—Broken Hill	32	1	3	Fixed Deposit—Bank of Australasia ...	400	0	0
Balance	493	3	2	Current Account—Bank of Australasia ...	27	9	2
	£579	3	2		£579	3	2

STATEMENT OF RECEIPTS AND EXPENDITURE FOR THE YEAR ENDED 31ST DECEMBER, 1906.

RECEIPTS.

To Balance, 1905				£22	11	10
Entrance Fees				27	6	0
Members' Subs., 1904	£6	6	0			
" " 1905	50	8	0			
" " 1906	357	0	0			
" " 1907	25	4	0			
				438	18	0
Bank Exchange				2	3	0
Sale of Publications				9	3	6
Interest on Deposit				13	0	0
Advertisements				15	0	0
				£528	2	4

EXPENDITURE.

By Printing, Stationery, &c.				£21	6	9
Publications—"Proceedings"—						
Vol. II., No. II.	4	17	6			
Vol. II., No. III.	28	16	0			
Vol. II., No. IV.	42	11	3			
Vol. III., No. I.	15	12	9			
"Transactions"-Vol. XI.	66	8	6			
"The Mount Lyell Mining Field"	73	19	2			
				232	5	2
Postages, Telegrams, &c.	£40	11	11			
Bank Exchange	4	9	0			
Freight, Transport, &c.	2	11	0			
Salaries, &c.	161	5	0			
Audit Fees	1	1	0			
				209	17	11
Prizes				12	0	0
Meeting Expenses—Adelaide-Broken Hill				19	6	10
Miscellaneous				5	16	6
Balance				27	9	2
				£528	2	4

MELBOURNE, 30*th January*, 1907. Audited and found correct. L. BYRON MORRES, F.I.A.V., *Auditor*.

MINUTES OF MEETINGS.

OF THE EXECUTIVE COMMITTEE.

(Abstract.)

AUGUST 27TH, 1906—12.30 P.M.

The balances to the credit of the Institute were reported as £81 on current account and £400 on fixed deposit.

Accounts to the amount of £43, including £15 for printing, were passed for payment.

The following nominations as members were received, *viz.*, Messrs. T. H. Davies, J. J. Gillio, W. C. W. Pearce and J. C. Moulden.

The Mining and Geological Institute of India was placed on the exchange list.

Messrs. H. H. Schlapp and A. S. Kenyon were appointed examiners of papers received from students competing for prizes offered by Messrs. A. T. Danks, Donald Clark and A. E. Kitson.

Resolved that circulars containing conditions under which prizes will be offered by the Institute during 1906 be issued at once. Papers to be returnable by 31st December, 1906.

OCTOBER 8TH, 1906—12.30 P.M.

The balances to the credit of the Institute were reported as £90 on current account and £400 on fixed deposit.

Accounts to the amount of £66, including £38 for printing, were passed for payment.

The following nominations as members were received, *viz.*, Messrs. T. C. Groom, R. Hamilton, W. B. Kenyon and W. E. Simpson.

The date for holding the Annual Meeting was fixed for 31st January.

Mr. G. A. Richard's resignation as local correspondent for Queensland was accepted.

Other routine business was transacted.

NOVEMBER 19TH, 1906—12.30 P.M.

The balances to the credit of the Institute were reported as £76 on current account and £400 on fixed deposit.

Accounts to the amount of £24 were passed for payment.

Ballot papers for the election of Members and Associate Members issued on the 18th September, 1906, were examined and the result showed that the following were elected, *viz.*, Members—Erwin Basedon, A. A. Boyd, J. Carthew, T. H. Davies, N. V. Grierson, R. Hamilton, C. G. Hylton, J. A. M. Lindsay, A. N. Macnicol, W. A. Maddern, J. C. Moulden, W. C. W. Pearce, W. E. Simpson, G. Ulrich and G. W. Wright. Associate Members—W. L. Cleland. E. T. M. Garlick, J. J. Gillio, H. L. Langford and H. M. Scruby.

The following were elected Students—Messrs. E. C. Dyason, E. Barkley, J. White, W. J. Hile and S. S. Strutt.

Nominations of R. Stoddart as a Member and F. Heyne and L. M. Downing as Associate Members were received.

DECEMBER 17TH, 1906.—12.30 P.M.

The balances to the credit of the Institute were reported as £94 on current account and £400 on fixed deposit.

Accounts to the amount of £113, including £61 for printing and £19 secretarial expenses, Adelaide meeting, were passed for payment.

Mr. L. Byron Morres was appointed Auditor for 1906 accounts.

Nominations of Messrs. Harvey S. Warde, Lloyd L. Paynter and William G. Williams were received.

Mr. John Atkinson was elected a Student-Member.

The report of the examiners of papers received from Students competing for the Danks, Clark and Kitson prizes was received and the prizes awarded as follows:

1st prize—(the gift of Mr. A. T. Danks)—
F. W. Sewell, Bairnsdale School of Mines.

2nd prize—(the gift of Mr. Donald Clark)—
A. D. Galloway, Ballarat School of Mines.

Special prize—(the gift of Mr. A. E. Kitson)—
H. C. Richards, Melbourne University.

FEBRUARY 11TH, 1907—1 P.M.

The balances to the credit of the Institute were reported as £186 on current account and £400 on fixed deposit.

Accounts to the amount of £72, including £43 for printing were passed for payment.

Messrs. H. H. Schlapp, A. H. Merrin and H. Herman were appointed a Publication Committee for 1907.

Mr. A. T. Danks's offer to donate a sum of ten guineas for a student's prize essay in connection with the next Institute excursion was accepted with thanks.

Nominations as follows were received:—Members—Messrs. D. H. Fisher, C. H. Humphreys, A. J. Hunter, J. Leahy, B. G. Patterson, E. J. J. Rodda, W. H. Schmidt and L. A. Westcott. Associate Member—Mr. S. Fry.

The following were elected Students:—Messrs. A. P. Penman, A. W. Sherritt and J. C. Coldham.

MARCH 11TH, 1907—12.30 P.M.

The balances to the credit of the Institute were reported as £148 on current account and £400 on fixed deposit.

Accounts to the amount of £16 were passed for payment.

Nominations of Mr. A. H. P. Moline as a Member and Mr. Ed. Edwards, jun., as an Associate Member, were received.

A list of members in arrears was submitted and it was resolved that these members receive notice in accordance with Rule 6.

The "Canadian Mining Journal" was placed on the exchange list.

APRIL 15TH, 1907—12.30 P.M.

The balances to the credit of the Institute were reported as £158 on current account and £400 on fixed deposit.

Accounts to the amount of £26, including £5 for printing, were passed for payment.

The nomination of Mr. Reginald F. Barker was submitted.

Mr. C. F. M'Master was elected a student.

Reports by the "Weights and Measures" and "Mesh" Committees as adopted by the Central Standardisation Committee of the Institution of Mining and Metallurgy were considered, and it was decided that copies be forwarded to members of Council.

Other routine business was transacted.

OF THE COUNCIL.

FEBRUARY 11TH, 1907—12.30 P.M.

An Executive Committee, consisting of members of the Council resident, or, at time of meeting, being in Melbourne, was appointed.

Mr. D. L. Stirling was appointed Secretary.

Mr. T. F. Brennan was appointed Honorary Auditor for 1907.

NOTICES.

The rooms of the Institute are open from 9.30 A.M. to 10 P.M. (Saturday, noon) daily, except Sundays and Public Holidays.

NEW ROOMS.

The Institute now occupies new rooms at 57-59 Swanston Street. The rooms are very convenient and complete and comprise a large reading room, lecture room and library (Telephone, No. 125 City) and writing materials. A Committee room and the Secretary's office are adjoining. The rooms are open daily and the lift in attendance from 9.30 a.m. to 10 p.m.

FIRST ORDINARY MEETING 1907.

It is probable that the First Ordinary Meeting, 1907, will be held at Launceston, Tasmania. Definite arrangements will be communicated to members shortly. The meeting will include excursions to places of mining and engineering interest in the North Eastern Part of Tasmania. The usual reductions in travelling rates will be obtained.

PUBLICATIONS

The following papers appear in this number:—"Bromo-Cyaniding of Gold Ores," by E. W. Nardin; "The Refining of Base Bullion at Port Pirie and Treatment of By-Products," by Brian B. Bayly; and "The Mining and Metallurgy of Copper, Silver, Lead and Zinc as Seen During the Excursion of the Australasian Institute of Mining Engineers, May, 1906," by Francis W. Sewell. Members are earnestly requested to forward papers descriptive of works, treatment, costs, &c., and to contribute discussions or additional information on those already published.

MEMBERSHIP.

The following applications for admission as Members and Associate Members have been received and are submitted for confidential report:—As Members—REGINALD F. BAKER, Leigh Creek, S.A.; ALFRED C. DUNN, Broken Hill, N.S.W.; BERTRAM

B. **Fabian**, Koorboora, Queensland; **Albert E. Fryar**, Rockhampton, Queensland; **Cecil G. Ryan**, Bradshaws Creek, Tas.; **Joseph W. Woodburn**, Mount Morgan, Queensland. As Associate Members:—**Brian B. Bayly**, Chiltern, Vic.; **Harold C. Kingsbury**, Irvinebank, Queensland; **W. Keverall M'Intyre**, Hobart, Tas.; **Hubert Rutter**, Melbourne, Vic.; and **Victor J. Waine**, Charters Towers, Queensland.

The following were elected on 31st May, 1907. Members:—**D. H. Fisher**, **T. Clive Groom**, **C. H. Humphreys**, **A. J. Hunter**, **W. Bedford Kenyon**, **John Leahy**, **A. H. P. Moline**, **B. G. Patterson**, **E. J. J. Rodda**, **W. H. Schmidt**, **R. Stoddart**, **Harvey S. Warde**, **L. A. Westcott**, and **Wm. G. Williams**. Associate Members:—**Ed. Edwards** and **S. Fry**. Associate:—**Frederick Heyne**.

STANDARDISATION.

The following copies of Reports by the "Weights and Measures" and "Mesh" Sectional Committees of the Inst. of Mining and Metallurgy are submitted to members in accordance with the circular and form just issued. Members are invited to forward replies to questions on the form (posted under separate cover), and to forward, in the form of a paper for the Proceedings, any data or particulars that may be of interest to metallurgists or induce discussion—

COPY OF THE REPORT OF SECTIONAL COMMITTEE "A"—WEIGHTS AND MEASURES, as adopted by the Central Standardisation Committee, 20th December, 1906, and submitted to the Council, 16th January, 1907.

TO THE CHAIRMAN AND MEMBERS OF THE CENTRAL STANDARDISATION COMMITTEE.

GENTLEMEN,—

The Weights and Measures Standardisation Committee submit the following recommendations for adoption by the Central Committee:—

1. That the word "ton" shall represent a weight of 2,000 lb. avoirdupois ($29{,}166\dot{6}$ oz. troy); that the use of the terms "cwt." and "qrs." be abandoned, and that fractions of a ton be expressed either in pounds or in decimals of a ton.

2. That the "miner's inch" be understood to mean a flow of 1·5 cub. ft. of water per minute.
3. That the word "gallon" be understood to mean the Imperial gallon of 10 lb.
4. That all temperatures be expressed in degrees Centigrade.
5. That gold and silver returns be expressed in terms of fine gold and silver, and not as "bullion."
6. That gold contents of ores, &c., be expressed in money values as well as in weights; and that in this connection the standard value be taken at 85 shillings, or $20·67 U.S. currency, per troy ounce fine gold.

They also suggest that the following questions be appended to any Memorandum that may be issued to the Members of the Institution embodying the above definitions :—

(*a.*) Do you consider the general adoption of the Metric system of weights and measures to be feasible in mining and metallurgical work, or would this in your opinion lead to undue dislocation ?

(*b.*) Have you any suggestion to make as to weights and measures, other than those already dealt with, which require exact definition ?

The question of the adoption of Metric standards has engaged the attention of the Committee, as the present movement in that direction in other branches of industry rendered it, in their opinion, inadvisable to ignore it entirely.

The feeling of the Committee, however, is that, while the question should be brought to the attention of the members, it would at present be quite inexpedient to advocate the general adoption of the Metric system in mining and metallurgical work.

On the other hand, an attempt to decimalize existing weights and measures is considered to be a step in the right direction, and as tending to bring about a desirable simplification.

I am, Gentlemen, yours obediently,

(Signed) H. LIVINGSTONE SULMAN,
Chairman of Sectional Committee "A"—Weights and Measures.

13th December, 1906.

COPY OF THE REPORT OF SECTIONAL COMMITTEE "C"—MESH, as adopted by the Central Standardisation Committee, 20th December, 1906, and submitted to the Council, 16th January, 1907.

TO THE CHAIRMAN AND MEMBERS OF THE CENTRAL STANDARDISATION COMMITTEE.

GENTLEMEN,—

I have pleasure in submitting the Report of Sectional Committee "C"—Mesh, on the questions referred to them for consideration.

The Committee unanimously recommend the adoption of the following as "The I.M.M. Standard Laboratory Screens":—

Mesh or apertures per linear inch.	Diameter of Wire.	Aperture.	Screening Area.
5	0·1 inch.	0·1 inch.	25·00 per cent.
8	0·063 ,,	0·062 ,,	24·60 ,,
10	0·05 ,,	0·05 ,,	25·00 ,,
12	0·0417 ,,	0·0416 ,,	24·92 ,,
16	0·0313 ,,	0·0312 ,,	24·92 ,,
20	0·025 ,,	0·025 ,,	25·00 ,,
25	0·02 ,,	0·02 ,,	25·00 ,,
30	0·0167 ,,	0·0166 ,,	24·80 "
35	0·0143 ,,	0·0142 ,,	24·70 ,,
40	0·0125 ,,	0·0125 ,,	25·00 ,,
50	0·01 ,,	0·01 ,,	25·00 ,,
60	0·0083 ,,	0·0083 ,,	24·80 ,,
70	0·0071 ,,	0·0071 ,,	24·70 ,,
80	0·0063 ,,	0·0062 ,,	24·60 ,,
100	0·005 ,,	0·005 ,,	25·00 ,,
150	0·0033 ,,	0·0033 ,,	24·50 ,,
200	0·0025 ,,	0·0025 ,,	25·00 ,,

NOTE.—Owing to difficulties in wire drawing and in the weaving of wire cloth, absolute accuracy to the fourth place of decimals of an inch is unattainable with uniformity of 25 per cent. of screening area; but the above table is so near to theoretical perfection, and the unavoidable irregularities of screening tests themselves are so wide, that any inaccuracies in the table would be absolutely immaterial in practice. It is not possible to weave the 200-mesh screen except in what is known as "twilled" or double wire.

The advantages of the series are:—

1. That a definite ratio and a corresponding arithmetical progression of both aperture and mesh is secured.
2. That by adopting a screening area of 25 per cent., the wires are absolutely "locked" in position, thereby preventing shifting and consequent irregularity in the size of aperture.

3. That the ratio between wire and aperture in all meshes being constant, the angle of taper of the hole is also constant.

The Committee append a Memorandum explaining in some detail the grounds and arguments upon which their recommendations are based, and they desire to add that these recommendations are the result of the most careful consideration of the views and suggestions of users of screens in all parts of the world, and of the views of British and American Manufacturers.

As regards the more exact definition of the word "Slimes," the Committee suggest :—

1. That material coarser than 150 mesh be described as "Sand," coarse or fine.
2. That material passing 150 mesh but settling in seconds in a inch column of water be described as "Meal," and—
3. That material settling more slowly in water be described as "Slimes."

(Signed) WALTER M'DERMOTT,
Chairman of Sectional Committee "C"—Mesh.

17th December, 1906.

Appendix to the Report of Sectional Committee "C"—Mesh.

On 3rd August, 1905, the Mesh Committee issued a printed circular to members of the Institution and others. Replies to the questions were received from over 100 Engineers.

There was practical unanimity as to the first three questions, from which fact the Committee considered it to be established—

1. That it is desirable to introduce a uniform system of laboratory grading by screen sizes.
2. That such uniformity can be secured by all engineers using an agreed standard series of screens in the laboratory.
3. That in giving their results engineers should state whether screening was done wet or dry.

Assuming the adoption of a standard series of screens, which makers of wire cloth could supply, there was practical unanimity as to the advantages of retaining the description of the various sizes by "*mesh per linear inch*," for the following reasons :—

Those who objected to the use of the term "mesh" based their objections on the fact that in the past it had not denoted the size of aperture; but as the whole object of establishing a laboratory standard would be to fix definitely a certain aperture for a certain mesh, it would be simply a matter of convenience of terms as to whether a screen were described as a certain "mesh," or with apertures expressed in decimals. There could be little doubt that in practice engineers would continue to use the term "mesh" by choice, because it is shorter, really descriptive, easily remembered, and precisely as accurate as, because interchangeable with, a certain size of aperture. An agreed standard table giving the size of aperture, in decimals of an inch, for each standard mesh, would obviate any confusion. The suggestion of one or two engineers that in future aperture should always be specified alone would be unworkable in many cases, and would not necessarily ensure accuracy or uniformity. Micrometer measurements of fine wire, when once woven and crimped are not always to be relied on, and uneven weaving would in practice vitiate conclusions as to size of aperture much more than in the case of the use of standard screens specially prepared for laboratory use. The dependence on every individual who reports screening tests to verify the measurement of such screens as he happens to have by means of a micrometer, even if advisable or practicable, would produce results of grading that could not easily be made comparable with results obtained elsewhere. The attempt to use a standard of apertures in arithmetical ratio, accurate to two or three places of decimals, is pronounced impracticable by wire cloth makers; and in any case would only prove to be the introduction of a new arbitrary scale of grading, differing greatly from past practice, and offering no advantage over an agreed standard set of screens such as are now advocated.

The report of the Transvaal Chamber of Mines "Sub-Committee on the Standarization of Battery Screening" covers a larger field than the Mesh Committee of the Institution have at present in view, and enters on the question of large-scale manufacture of wire cloth and screens and their use in actual crushing operations. As regards the description of screens in mills, the Mesh Committee do not propose for the moment to put forward any suggestions. There are difficulties to be considered from the fact that Managers of Mines are likely to be guided by personal opinions of expediency in connection with the kind of ore, type of crusher, cost of material, and subsequent processes of treatment. It is well known that different ores, different quantities of water, wet or dry crushing, and different crushers, all give varying characters of discharge through exactly the same screen. The Committee therefore recommend, in the first instance, the establishment of one uniform standard method of expressing grading analysis of all mill products in the laboratory, which would enable accurate comparisons to be made.

Only comparative accuracy can be attained in the description of *mill* screens made in commercial quantities, owing to the fact that, both in the drawing of the wire and in the weaving, considerable irregularity always occurs.

There is practical unanimity in approving the adoption of a standard table which, while not departing much in the size of aperture from what is already associated with certain commonly used meshes, would secure a regular arithmetical ratio between mesh and aperture. In considering the table embodied in the Committee's report the following points should be kept in view :—

There must be limits both at the coarse and fine ends of the table, to be determined by consideration of utility, common practice, and requirements of manufacture. The number of sizes between these limits should be reasonable, for it is not desirable to introduce excessive refinements and a large number of screens in the laboratory; and it is not necessary to ask makers to manufacture in small quantities with accuracy, and keep in stock, a number of standards which the great majority of engineers will never use in practice. The engineer who requires special refinements in a particular case can use any kind of screen and by giving its aperture enable comparison to be made with the nearest standard mesh. Grading by screens can never be exactly scientific as a measure of the size of particles, and therefore it is useless to pretend to unnecessary refinements; and the usual practical requirements in ore treatment may be perfectly well secured in the classification of particles by a comparatively few screen sizes, as hitherto. As an illustration of the futility of taking screen grading as an exact scientific measurement of particles, it is only necessary to point out that the actual size of the maximum particles passing through any given screen varies with different ores, different methods of grinding, and different methods of using the screens.

After consultation with manufacturers of wire cloth, it appears that there would be no difficulty in getting makers to weave accurately the small quantities of the standard screening recommended by the Committee to supply the demands of all engineers for laboratory purposes.

There seems to be a general opinion that material passing the finest screen of any standard series should be divided into two classes, and not be described under the one heading of "Slimes." Various suggestions have been made as to the method of division and as to the terms to be used for describing the three degrees of fineness of particles. The general opinion seems to be that all material coarser than 150 mesh should be described as "Sand"—coarse or fine, if preferred; that the portion passing 150 mesh but readily settling in water should be described by some such term as "Meal;" and that the term "Slimes" should be limited to particles which settle slowly in water. It may be difficult to get any general agreement as to the division between "Meal" and "Slimes"; and even if this be made arbitrarily, it will remain a fact that two materials having the same settling speeds may not be of the same size of particles, character, or permeability. Some such rough division of the two classes would, however, tend to greater accuracy and uniformity of description than is usual at present, and it is suggested by the Mesh Committee that particles passing 150 mesh but settling in   seconds in a   inch column of water be described as "Meal."

COPY OF LETTER FROM MR. H. L. SULMAN.

LONDON, E.C., 6th February, 1907.

TO THE CHAIRMAN, THE MESH SUB-COMMITTEE,
THE INSTITUTION OF MINING AND METALLURGY.

DEAR SIR,—

I beg to enclose herewith *a series of photo-enlargements of carefully-woven steel wire screening. The samples were submitted to me by a well-known firm of manufacturers for photo-micrographic purposes, and represent first-class samples of wire screening material. Inspection will show that a wide difference exists between the area of the largest and smallest apertures in each sample investigated, and the photos. fully confirm the conclusions of your Committee as to the inutility of closer gradations in mesh range than are contained in their recommendations.

It will be evident that the errors introduced in weaving such screens far outweigh any hypothetical accuracy possible of attainment by mere exactitude in gauge of the wire employed, and, indeed, of any minute requirements as to calculated aperture.

I have compared the largest and smallest mesh apertures obtained in each of these photos., and find the error to vary between 10 per cent. and over 20 per cent., taking the area of the largest aperture in each case as 100. These screen samples are of new material, and have not been used; the error would of course be likely to increase considerably after use.

It will also be noticed that the various linear dimensions of individual apertures vary considerably even in cases of closely approximating aperture area.

I am, dear Sir, yours truly,

H. LIVINGSTONE SULMAN.

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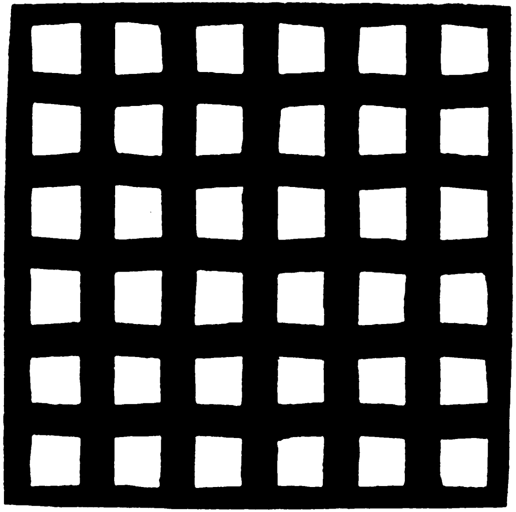
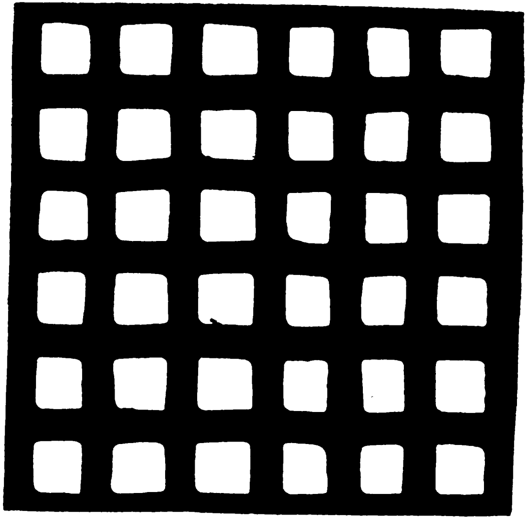
* See the two examples and comparative table re-produced on pages lxii and lxiii.

PHOTO-ENLARGEMENTS OF STEEL WIRE SCREENING.

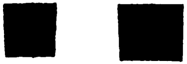
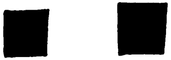
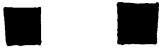
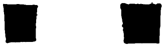
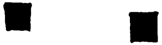
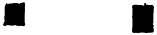


40 × 40 Mesh (Magnified 18 diameters).



60 × 60 Mesh (Magnified 26 diameters).

COMPARATIVE TABLE.

Smallest and largest apertures.	Description of Sieve.					Difference in area between largest and smallest meshes
	Mark.	Mesh.	Wire.*	Aperture*	Screening Area.	
	"LWA"	24 x 25	·018	·0236 ·022	31·15%.	21%.
	"AKB"	30 x 30	·0136	·0197	34·92%.	16·3%.
	"AMA"	36 x 34	·0124	·0153 ·017	31·83%.	11%.
	"WKB"	40 x 40	·0108	·0142	32·26%.	13·4%.
	"DKA"	50 x 50	·0092	·0108	29·16%	22%.
	"MKA"	60 x 60	·0076	·0090	29·16%	20%.

(* In decimals of an inch).

Comparison of largest and smallest apertures when uniformly magnified to 13 diameters.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY

From 1st July, 1906, to 30th June, 1907.

Australian Mining Standard	weekly	Melbourne
Iron and Coal Trades Review	weekly	London
Mining Journal	weekly	London
Page's Weekly	weekly	London
Engineering and Mining Journal	weekly	New York
Engineering Record	weekly	New York
Mining and Scientific Press	weekly	San Francisco
Mines and Minerals	monthly	Scranton, Pa.
Indian Engineering	weekly	Calcutta
Chamber of Mines of Victoria :	Monthly Mining Report	Melbourne
South African Engineering	weekly	London
The West Australian Mining, Building and Engineering Journal	weekly	Perth
Royal Society of Victoria : Proceedings	Vol., IX., Part 1,	Melbourne
Victorian Department of Mines :		
Bulletins, Nos. 19 to 22		Melbourne
Memoirs, Nos. 3 and 4		Melbourne
West Australian Department of Mines :		
Mining Statistics—quarterly		Perth
Bulletins, Nos. 22 to 25		Perth
Chamber of Mines of West Australia :	Journal—monthly	Kalgoorlie
South Australian Department of Mines :		
Report on the Northern Territory		Adelaide
South Australian School of Mines :		
Annual Report, 1906		Adelaide
University of Adelaide : Calendar, 1907		Adelaide
Moonta School of Mines : Annual Report, 1906		Moonta
New Zealand Department of Mines :		
Bulletin No. 1, Geological Survey		Wellington
Mines Record—monthly		Wellington
The New Zealand Mining Handbook, 1906		Wellington
New Zealand Institute :		
Transactions and Proceedings	Vol. XXXVIII.,	Wellington
Report of the Inspection of Machinery Department, 1905-6		Wellington

- Tasmanian Department of Mines :
  - Progress of the Mineral Industry—quarterly — Hobart
- Queensland Department of Mines :
  - Annual Report, 1906 - - - Brisbane
  - Mining Journal—monthly - - - Brisbane
  - Geol. Sur. Report, No. 201 - - Brisbane
- Royal Society of Quensland : Proceedings, Vol. XIX , Part II., Brisbane
- Society of Chemical Industry :
  - Journal—bi-monthly - - - London
- Institution of Mechanical Engineers :
  - List of Members ; Proceedings, No. 3, 1906 — London
- Iron and Steel Institute : Papers, May, 1907 - London
- Institution of Mining Engineers :
  - Transactions, Vol. XXVIII., Part 6 ; Vol. XXIX., Part 6 ; Vol. XXX,, Part 5 ; Vol. XXXI., Parts 2 to 5 ; Vol. XXXII., Parts 1, 3 and 4 - Newcastle
- Geological Society : Quarterly Journal - Vol. LXIII., Part 1, London
- Geologists' Association : Proceedings - Vol. XIX., Part 9, London
- Institute of Chemistry of Great Britain and Ireland : Proceedings - - - Part II., 1907, London
- Royal Dublin Society :
  - Economic Proceedings - - - Vol. I., Part 8, Dublin
  - Scientific Proceedings - - Vol. XI., No. 12, Dublin
- The Cruise of the Neptune—A. P. Low - Ottawa
- Geological Survey of Canada :
  - Annual Report - - - Vol. XV., 1902-3, Ottawa
  - Palæozoic Fossils - - - Vol. III., Part IV., Ottawa
- British Columbia Department of Mines :
  - Annual Report, 1905 - - - Victoria
- Nova Scotia Institute of Science :
  - Proceedings and Transactions - Vol. XI., Part 2, Halifax
- Cape Colony Department of Agriculture :
  - Tenth Annual Report of the Geological Commission - - - - Cape Town
- Geological Survey of India :
  - Records, Vol. XXXIII., Part 4 ; Vol. XXXIV., Parts 1, 2 and 3 ; Vol. XXXV., Part 1 - - - Calcutta

Mining and Geological Institute of India :
Transactions - - - Vol. I., Parts 1 to 4, Calcutta
Mysore Geological Department : Records - Vol. V., Bangalore
Memoirs - - - - Vol. III., Part I., Bangalore
Chemical, Metallurgical, and Mining Society of South Africa : Journal - Vol. VII, Nos. 4 to 10, Jonannesburg
Transvaal Chamber of Mines :
Monthly Analysis of Gold Production - Johannesburg
United States Geological Survey :
Annual Report, 1904-5 - - - Washington
Mineral Resources, 1904.
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Professional Papers, Nos. 44, 45, 47, 48, Parts 1, 2 and 8 ; 49, 50, 51 and 55.
Bulletins, Nos. 265, 269, 272 to 275, 277, 278, 280 to 285, 288 to 293, 298, 301.
Water Supply Papers, Nos. 153 to 160, 162 to 165, 167 to 170, 172 to 175, 179 to 181, 186.
American Institute of Mining Engineers :
Bulletin—bi-monthly - - - New York
American Society of Civil Engineers :
Transactions - - Vols. LVI. and LVII., New York
Columbia University : Scientific Papers - New York
Missouri Bureau of Geology and Mines :
Geology of the Granby Area, Geology of Moniteau County, Biennial Report of State Geologist - - - Jefferson City
Maryland Geological Survey : Vol. V. - - Baltimore
Smithsonian Institution : Annual Report, 1904 Washington
Colorado Scientific Society :
Proceedings, pp. 50-70, 103-166.
Field Columbian Museum :
Publications, Nos. 107, 109-114 - - Chicago
Iowa State College :
Iowa Engineer—bi-monthly - - Ames
Franklin Institute :
Journal, Vol. CLXI., No. 6 ; Vol. CLXII., Nos. 1 to 6 ; Vol. CLXIII, Nos. 1, 2 and 5 - - - - Philadelphia

Massachusetts Institute of Technology:
Technology Quarterly - - Vol. XIX., Nos. 2 and 3, Boston
Société Géologique du Nord: Annals - - Vol. XXXIV., Lille
Annales des Mines: Vol. IX., Parts 1 to 12;
Vol. X., Parts 1 to 12, Vol. XI., Parts 1, 2 Paris
Les Mois Scientifique: Monthly - - Paris
Société des Ingénieurs Civils de France:
Memoirs, 6th Series, 59th Year, Nos. 3 to 12; 6th Series, 60th Year, Nos. 1, 2 and 3 - - - - Paris
Foldtani-Kozlony:
Proceedings - - Vol. XXXVI., Nos. 3 to 12, Buda-pest
Montan Zeitung—weekly - - - Graz
Musée Teyler: Archives, Series II., Vol. IX., Part 4; Vol. X., Parts 1 to 3 - - Haarlem
University of Upsala: Bulletin - Vol. VII., Nos. 13 and 14, Upsala
Sverige Geologiska Undersökning:
Transactions - - Series C, Nos. 197 to 199, Stockholm
Real Academia des Ciencas:
Memoirs - - - Vol. IV.; Vol. V., Nos. 1 to 4, Madrid
Memoirs - - - - - Vol. XXIV., Madrid
Institut Grand-Ducal De Luxembourg:
Archives - - - - Parts 1 and 2, Luxembourg
Societá Geologica Italiana: Bulletins, Vol. XXIV., Nos. 1 and 2; Vol. XXV., Nos. 1 to 3 - - - - - Roma
Societá degli Ingegneri:
Annals - - - - Vol. XXI., No. 1, Roma
Bulletins - - - - Vols. XI. and XII., Roma
Accademia Della Sienze - - New Series, Vol. IX., Bologna
Comité Géologique: Memoirs - - Nos. 3, 18 to 20, St. Petersburg
Bulletins - - Vol. XXIII., Nos. 7 to 9, St. Petersburg
Instituto Geologico de Mexico:
Bulletin, No. 22 - - - - Mexico
Cuerpo de Ingenieros de Minas:
Bulletins, Nos. 32 and 33, 41, 44 to 46 - Lima
Museo Nacional de Buenos Aires:
Annals - - - Series III., Vol V., Buenos Aires

RECENT ARTICLES ON MINING MATTERS.

(1st July, 1906, to 30th June, 1907.)

NOTE.—*This list is prepared for the purpose of placing before members the titles of the more important papers appearing in the usual publications concerned with mining engineering, due regard being had to Australasian requirements.*

LIST OF PUBLICATIONS.

References are given by the number prefixed to each publication in the attached list. Wk., weekly; mth., monthly.

(1) *The Australian Mining Standard*, Melbourne, Victoria, wk., 6d.

(2) *The Queensland Government Mining Journal*, Brisbane, mth., 6d.

(3) *The New Zealand Mines Record*, Wellington, N.Z., mth., 1s.

(4) *The Mining Journal*, London, E.C., wk., 6d.

(5) *Page's Weekly*, Surrey-street, London, W.C., wk., 6d.

(6) *The Engineering and Mining Journal*, New York, wk., 15c.

(7) *Mines and Minerals*, Scranton, Pa., U.S.A., mth., 20c.

(8) *Mining and Scientific Press*, San Francisco, Cal., wk., 10c.

(9) *Annales des Mines*, Paris, France, mth.

(10) *Publications*, Department of Mines, Melbourne, Victoria.

(11) *Publications*, Department of Mines, Sydney, New South Wales.

(12) *Publications*, Department of Mines, Adelaide, South Australia.

(13) *Publications*, Department of Mines, Brisbane, Queensland.

(14) *Publications*, Department of Mines, Perth, Western Australia.

(15) *Publications*, Department of Mines, Hobart, Tasmania.

(16) *Publications*. Geological Survey, Canada, Ottawa, Ontario.

(17) *Publications*, Bureau of Mines, Toronto, Ontario.

(18) *Publications*, Geological Survey, India, Calcutta.

(19) *Publications*, Geological Survey, U.S.A., Washington.

(20) *Publications*, Geological Survey Alabama, Montgomery, Ala.

(21) *Publications*, California State Mining Bureau, Sacramento, Cal.

(22) *Reports* Aust. Assoc. Adv. Science, Sydney, New South Wales.

(23) *Transactions and Proceedings*, New Zealand Inst., Wellington, New Zealand.

(24) *Quarterly Journal*, Geological Society, London.

(25) *Transactions*, Inst. Mining and Metallurgy, London, E.C.

(26) *Transactions*. Fed. Inst. Min. Eng., Newcastle-upon-Tyne, Eng.

(27) *Journal*, Canadian Mining Inst., Ottawa Ontario.

(28) *Journal*, Chem., Min., and Met. Soc. of S.A. Johannesburg, Transvaal.

(29) *Transactions*, Am. Inst. of Min. Eng., New York City.

(30) *Proceedings*, Colorado Scientific Soc., Denver, Col.

(31) *Journal*, Franklin Inst., Philadelphia, Pa.

(32) *Technology Quarterly*, Mas. Inst. Tech. Boston, Mass., 75c.

(33) *Transactions*, Am. Soc. C.E., New York City.

(34) *Bulletins* Société des Ingénieurs Civils Paris.

(35) *Mining Magazine*, 120 Liberty-street, New York, mth., 20c.

(36) *Publications*, Iron and Steel Institute London.

(37) *Proceedings*. Inst. of Mech. Eng., London.

(38) *Publications*, Field Columbian Museum, Chicago, U.S.A.

(39) *Journal*, Mining Society of Nova Scotia, Halifax, N.S.

(40) *Transactions*, Mining and Geological Institute of India, Calcutta.

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COAL MINING.

Report on Coal-Testing Plant. E. W. Parker, J. A. Holmes, M. R. Campbell. (19) Prof. Papers, No. 48.
The Coal Measures near Townsville. L. C. Ball. (2) Aug., 1906.

DESCRIPTIVE.

The Washoe Plant of the Anaconda Copper Co. in 1905. L. S. Austin. (29) Bull. No. 10.
Methods of Mining, Hauling, and Screening at the Mines of the Aldrich Mining Co. at Brilliant, Ala. T. H. Aldrich, jr. (29) Bull. No. 10.
Mining Conditions in the Mountains of Chihauhau. J. B. Farish. (6) Feb. 2, 1907.
Mining Methods. W. R. Crane. (7) Serial commenced Aug., 1906.
The Auriferous Deposits and Mines of Menzies. H. P. Woodward. (14) Bull. No. 22.
The Laverton, Burtville, and Erlistoun Auriferous Belt. C. E. Gibson. (14) Bull. No. 24.
Some Mines in the Burnett District. B. Dunston. (2) July, 1906.
The Annan River Tinfield. W. E. Cameron. (2) Nov., 1906.
Some Goldfields of Cape York Peninsula. W. E. Cameron. (2) Dec., 1906.
The O.K. Copper Mine, North Queensland. R. L. Jack. (2) Jan., 1907.
Some Croydon Gold Mines. B. Dunston. (2) May, 1907.
Black Ridge, Clermont. L. C. Ball. (13) Pub. No. 201.
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The Lauriston-Drummond North Goldfield. W. Baragwanath. (10) Bull., No. 19.
The Berringa Goldfield. W. Baragwanath, jr. (10) Memoirs, No. 5.
Mining in Korea. J. H. Curle. (8) July 21, 1906.
Copper at Butte, Montana. A. H. Halloran. (8) Serial commenced Aug. 11, 1906.
Mining in South Africa. J. B. Pitchford. (8) Serial commenced Mar. 9, 1907.
Auriferous Tracts in Southern India. J. M. Maclaren. (18) Vol. XXXIV., Part 2.

DREDGING.

Gold Dredging in the Urals, with Notes on Dredging in Siberia. W. H. Shockley. (29) Bull. No. 10.
Control of Hydraulic Mining in California by the Federal Government. W. W. Hart. (33) Vol. LVII.
Hydraulic Mining in California. J. P. Hutchins. (6) May 19, 1906.
The Rehabilitation of Hydraulic Mining. J. P. Hutchins. (6) Nov. 10 and 17, 1906.
Placer Mining in the Klondike. J. B. Tyrrell. (6) Mar. 2, 1907.
Placer Mining in French Guiana. L. Delvaux. (6) Mar. 2, 1907.
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Placer Mining Methods. A. Carmichael (7) Jan., 1907.
Prospecting Dredging Ground. D. Weatherbe. (8) Oct. 20, 1906.

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Electric Winding for Main Shafts. W. C. Mountain. (26) Vol. XXXI., Part 4.
Electrical Mining Hoists. J. W. H. Hamilton. (6) Sept. 22, 1906.
Electrical Winding Plants. A. Gradenwitz. (6) Feb. 2, 1907.
Electrical Equipment. G. R. Wood. (7) Dec., 1906.

GEOLOGICAL.

Ore Deposits of the Silver Peak Quadrangle, Nevada. J. E. Spurr. (19) Prof. Papers, No. 55.
Geography and Geology of Alaska. A. H. Brooks. (19) Prof. Papers, No. 45.
Contributions to Economic Geology, 1905. S. F. Emmons and E. C. Eekel. (19) Bull. No. 285.
Corundum: its Occurrence and Distribution in the United States. J. H. Pratt. (19) Bull. No. 269.
The Formation of Coal. M. R. Campbell. (5) July, 1906.

Prospects of Obtaining Artesian Water in the Kimberley District. R. L. Jack. (14) Bull. No. 25.

Magmatic Separation in its Relation to the Genesis of Certain Ore Bodies. James Park. (23) Vol. XXXVIII.

Contact Metamorphism in its Relation to Certain Metalliferous Veins. James Park. (23) Vol. XXXVIII.

Thermal Activity in its Relation to the Genesis of Certain Metalliferous Veins. (23) Vol. XXXVIII.

On the *Rôle* of Metasomatism in the Formation of Certain Ore Deposits. James Park. (23) Vol. XXXVIII.

The Deposition of Mineral Matter from Aqueous Solutions in its Relation to the Filling of Cavities and Vein Fissures. James Park. (23) Vol. XXXVIII.

The Ballarat East Goldfield. J. W. Gregory. (10) Memoirs, No. 4.

Dikes. . J. F. Kemp. (8) Jan. 19, 1907.

MECHANICAL.

Turbine Water-wheel Tests and Power Tables. R. E. Horton. (19) Water Supply Papers, No. 180.

Practical Notes on Winding Ropes and Capels. E. Barraclough. (26) Vol. XXX., Part 5.

An Appliance for Automatically Stopping and Starting Mine Waggons. Prof. W. Galloway. (26) Vol. XXXII., Part 1.

A Diamond Hand-boring Machine. J. B. Thomson. (26) Vol. XXXII., Part 1.

A Rattan Exhaust Steam-driven Three-phase Haulage Plant. W. Maurice. (26) Vol. XXXII., Part 1.

Investigation on the Use of Steam in Gas Producer Practice. W. A. Bone and R. V. Wheeler. (36) Papers, May, 1907.

Induced Draught, with Hot Air Economizers for Steel Works and Blast Furnace Boilers. A. J. Capron. (36) Papers, May, 1907.

A Simple Rotary Distributor for Blast Furnace Charges. D. Baker. (29) Bull. No. 10.

Steam Consumption of a Modern Hoisting Plant. S. Powell, jr. (6) Sept. 15, 1906.

Skip Hoisting. S. A. Worcester. (6) Sept. 1, 1906.

Mine Timbering at Lake Superior. W. R. Crane. (6) Nov. 10, 1906.

Coke Oven Construction. W. M Judd. (6) Nov. 10, 1906.

Shaft-Sinking Through Water-Bearing Formations. E. M. Heriot. (6) Dec. 15, 22, and 29. 1906.

Centrifugal Pumps. R. C. Williams. (6) Feb. 16, 1907.

Diamond Drills. J. Humes. (6) Feb. 23, 1907.

Belt Elevators for Ore and Water. E. S. Wiard. (6) March 23, 1907.

Selection in Air Compressors. J. D. Cone. (7) October, 1906.

Handling Ore in Stopes. D. T. Williams. (7) Nov., 1906.

Shaft-Sinking. B. H. Dunshee. (7) Jan., 1907.

Some Characteristics of Coal as Affecting Steam Boilers. W. L. Abbott. (7) Feb., 1907.

Stoping Systems at Broken Hill. A. J. Moore. (7) May, 1907.

The Development of Blast Furnace Blowing Engines. D. E. Roberts. (37) July, 1906.

An Improved Method of Framing Square Sets. A. A. Steel. (8) Aug. 11, 1906.

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A Study in Refining and Overpoling Electrolytic Copper. Hofman, Hayden, and Hallowell. (29) Bull. No. 14.

Recent Progress in Metallurgy. A. E. Onterbridge, jr. (31) Vol. CLXII., No. 5.

Picking Belts. E. H. Messiter. (6) June 16, 1906.

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Ore-Milling in Wisconsin. (6) July 28, 1906.

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Argall, A. E., Gadag P.O., Bombay Presidency, India ..	1900
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Beamish, W. A., Cope's Creek Central T.D. Co. N.L., Tingha, N.S.W.	1904
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‖Beaumont, Edwin K., "Kenilworth," 1,716 Lewis-street, Lamington Heights, Kalgoorlie, W.A.	1899
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§Bridge, John M., Chapple-street, Broken Hill, N.S.W.	1907
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¶Broome, Geo. H., Romilly-street, Westport, N.Z.	1900
‖Browne, D. H., Great Northern Extended Consols G.M. Co., Rutherglen, Vic.	F.
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Burrows, Horace, 157 Queen-street, Melbourne, Vic.	1902
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Carlin, M. Henry, Pioneer Tin Mining Co., Bradshaw's Creek, Tas.	1898
Carthew, John, Broken Hill, N.S.W.	1906
Cautley, A. O., Belton Rectory, Great Yarmouth, England	1902
Challen, M. Bradbury, School of Mines, Daylesford, Vic.	1895
Chandler, Thomas, Emmaville, N.S.W.	F.
Channon, H. J., Wyalong, N.S.W.	1904
‖Chapman, R. W., University, Adelaide, S.A.	1901
Chapple, Albert J. M., Occidental Mine, Cobar, N.S.W.	1906
Chudleigh, Frederick, Wyalong, N.S.W.	1902
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Clarke, Huntley J., Lottah, Tas.	1901
‡Cleland, Wm. Lauder, Stannary Hills, *via* Cairns, North Queensland	1907
Coane, John M., 70 Queen-street, Melbourne, Vic.	1897
Cobbe, H. N. G., Box 251, Kalgoorlie, W.A.	1901
Cock, John, Chiltern, Vic.	1895
§Coldham, J. C., Broken Hill, N.S.W.	1907
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Coote, Frank J., Tongio West, Vic.	1905
Copland, M. O., c/o R. A. Tarlton, P.O. Box 468, Johannesburg, Transvaal	1904
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Courtney, C. F., Sulphide Corporation Ltd., c/o Gibbs, Bright and Co., Queen-street, Melbourne, Vic.	1901
Cunningham, John C., c/o Broken Hill South S.M. Co., Broken Hill, N.S.W.	1902
Daggar, H. James, The Associated Gold Mines of W.A. Ltd., P.O. Box 172, Kalgoorlie, W.A.	1901
Dalgleish, George, Norseman, W.A.	1904
Daly, H. J., 60 Queen-street, Melbourne, Vic.	1905
‡Danks, Aaron T., 391 Bourke-street, Melbourne, Vic.	1905
Davey, George, 34 Carlton Vale, Maida Vale, London W., England	1900
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Davies, R. H., Victorian Cornish Gold Mines Ltd., Daylesford, Vic.	1905
Davies, Thos. Hy., Bulong, W.A.	1906
‖Davies, W. A. T., Bulong, W. A.	1906
Davis, John, 54 Victoria-road, Hawthorn, Vic.	1900
‖Day, Percy R., Box 16, Kalgoorlie, W.A.	1898
Dean, A. L., Queenstown, Tas.	1905
‖Delprat, G. D., Broken Hill Proprietary Mine, Broken Hill, N.S.W.	1902
‖Dempster, H., Burnie, Tas.	F.
‖Denny, G. A., 564 Salisbury House, London, E.C., England	F.
‡Dibdin, Charles, Mount Morgan, Queensland	1902
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Donaldson, Robert J., Central Mine, Broken Hill, N.S.W.	1905
Donnelly, Alf., Murchison Goldfield, Day Dawn, W.A.	F.
Doolan, Thos. J., "Loretto," Kipling-street, St. Kilda, Vic.	1905
§Downing, L. M., The Copper Mine, Bundarra South, N.S.W.	1906
Dreyer, J. G., Central Mine, Broken Hill, N.S.W.	1902
Driffield, E. Carus, Queenstown, Tas.	F.
§Dyason, E. C., Trinity College, Melbourne, Vic.	1906

Names and Addresses.	Date of Election.
‖Dyson, Thos., Zeehan, Tas.	F.
‖Edmands, H. R., Menzies, W.A.	1901
Edwards, David, Parramatta Mine, North Yelta, S.A.	1902
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Edwards, Thos., Pyrites Works, Sebastopol, Vic.	1894
Elfverson, Wm., Perseverance Mine, Coolgardie, W.A.	1898
Elliott, James E., Goldfields Diamond Drilling Co. Ltd., 339 Collins-street, Melbourne, Vic.	1904
Evans, James H., Waihi Syndicate Ltd., Waihi, N.Z.	F.
Field, Edwin R., Daylesford, Vic.	1904
Fisher, D. H., Mount Morgan, Queensland	1907
Fisher, Hamilton, Sea Point Hall, Capetown, Cape Colony	1903
Ford, H. W., Boring Party, Marong, Vic.	F.
Fraser, J. C., Mercantile Chambers, Victoria-square, Adelaide, S.A.	F.
‡Fry, Sidney, Westport, N.Z.	1907
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Gardner, W. E., Block 10, Broken Hill, N.S.W.	1902
Garland, Chas. L., 154 Phillip-street, Sydney, N.S.W.	1898
‡Garlick, E. T. M., Hampden Mine, Cloncurry, Queensland	1906
Gilmour, Jas. L., Waihi, N.Z.	1904
‖Godfrey, J. R., "Graylings," Alma-road, St. Kilda, Vic.	1895
‖Gordon, Henry A., 28 His Majesty's Arcade, Auckland, N.Z.	1900
Grayson, L., Union Engineering Company, North-terrace, Adelaide, S.A.	F.
‖Grayson, Lawrence W., 414 Collins-street, Melbourne, Vic.	F.
Green, Leonard C., Technical College Laboratory, Brisbane, Queensland	1904
Greenway, T. J., 60 Brookman's Buildings, Grenfell-street, Adelaide, S.A.	F.
‖Gregory, J. W., The University, Glasgow, Scotland	1900
Grierson, Norman V., Higginsville, *via* Coolgardie, W.A.	1906
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Hack, Clem A., 63 Queen-street, Melbourne, Vic.	1905
Haig, J. M., Golden Link Consolidated Gold Mine, Kalgoorlie, W.A.	1902

Names and Addresses.	Date of Election
Hall, W. Leslie R., Surrey Chambers, St. George's-terrace, Perth, W.A.	1901
Hamilton, R., Great Boulder Proprietary Gold Mines Ltd., Kalgoorlie, W.A.	1906
Hancock, H. Ley, Box 94, Kalgoorlie, W.A.	1901
‖Hancock, H. Lipson, Moonta Mines, S.A.	F.
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‖Hancock, Leigh G., Burnie, Tas.	1902
Harris, C. M., Cosmopolitan Proprietary Ltd., Kookynie, W.A.	1901
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Haszard, N. F. J., Waihi, Auckland, N.Z.	1903
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‖Herman, H., 60 Queen-street, Melbourne, Vic.	1897
**Heyne, Frederick, Fairfield, Vic.	1907
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Hitchcock, W. E., Mathinna, Tas.	1904
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‖Hunter, Stanley B., Mines Department, Melbourne, Vic.	1894
Hylton, C. G., Broken Hill Proprietary Co., Broken Hill, N.S.W.	1906
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‖Jaffray, John W., Equitable Buildings, George-street, Sydney, N.S.W.	F.
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James, Caradoc, Golden Link Consolidated G.M, Kalgoorlie, W.A.	1894
‖Janitzky, Edward, "Pymble," North Sydney, N.S.W.	1897
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Johnston, W. H., Waihi, N.Z.	1904
‖Jolly, Leslie, Launceston, Tas.	1897
‖Jones, David, Waratah, Tas.	F.
Jones, Peter, Hillgrove, N.S.W.	1905

Names and Addresses.	Date of Election
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‖Kayser, H. W. Ferd., Launceston, Tas.	F.
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Kelly, Geo. W., 529 Little Bourke-street, Melbourne, Vic.	1898
‖Kenyon, A. S., Treasury Buildings, Melbourne, Vic.	1896
Kenyon, W. Bedford, Kalgoorlie, W.A.	1907
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‖Knox, Wm., 31 Queen-street, Melbourne, Vic.	F.
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Leahy, John, Broken Hill, N.S.W.	1907
Leggo, A. Victor, 22 Park-street, St. Kilda, Vic.	1898
Leggo, H. M., Associated G.M., Boulder, W.A.	1901
Lewis, James B., Lottah, Tas.	1906
Lindsay, John A., Broken Hill, N.S.W.	1906
Loosemore, Arthur F. M., Gympie, Queensland	1906
‖Lovely, W. H. C., Club Chambers, Phillip-street, Sydney, N.S.W.	F.
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Macartney, E. H. B., 53 Lewis-street, Kalgoorlie, W.A.	1901
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M'Lauchlan, R. C. H., 81 Grove-street, Balmain, Sydney, N.S.W.	1902
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MacLeod, W. A., Charters Towers, Queensland	1904
Macnicol, A. N., 31 Queen-street, Melbourne, Vic.	1906
Maddern, W. A., Junction North Mine, Broken Hill, N.S.W.	1906
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Mars, F. J., Blende-street, Broken Hill, N.S.W.	1902
Martin, Edward, South A.I. G.M. Syndicate, Gaffney's Creek, Vic.	1902
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May, A., Gawler Foundry, Gawler, S.A.	F.
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Milne, David, Broken Hill, N.S.W.	1902
Mixner, Alfred F., Pioneer G.M., Bright, Vic.	1900
Moline, Arthur H. P., Mt. Lyell Mining Co., Gormanston, Tas.	1907
‖Montgomery, A., Department of Mines, Perth, W.A.	F.
Moody, C. J., Lake View Consols G.M., Boulder, W.A.	1902
‖Morgan, Percy G., Geol. Survey Camp, Hokitika, N.Z.	1900
Morse, Phillip S., Sulphide Corporation Ltd., Cockle Creek, N.S.W.	1905
Morse, Willard S., Seaford, Del., U.S.A.	1901
‖Moss, Frank A., Kalgurli G.M., Kalgoorlie, W.A.	F.
Moulden, J. C., Sulphide Corporation Ltd., Cockle Creek, N.S.W.	1906
‖Nardin, E. W., Archer-street, Chatswood, N.S.W.	1898
Nicholas, C. E., Stannary Hills Mines, N. Queensland	1901
Nicolson, R. B., Ivanhoe Gold Corporation Ltd., Kalgoorlie, W.A.	1901
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Patterson, B. G., Mount Morgan, Queensland	1907
‖Paull, W. J., Charters Towers, Queensland	F.
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Pettit, Arthur E., 8 Old Jewry, London, E.C., England	1905
‡Philbin, Jas. P., Forest-street, Subiaco, W.A.	1906
Plant, E. H. T., Charters Towers, Queensland	1901
Poole, Wm., School of Mines, Charters Towers, Queensland	1905
‖Potter, J. M., Watsonville, Queensland	F.
‖Power, F. Danvers, 68½ Pitt-street, Sydney, N.S.W.	F.
†Raymond, Rossiter W., 99 John-Street, New York City, U.S.A.	1894
Read, H. Lawrence, 2,563 Washington-street, San Francisco, Cal.	1906

Names and Addresses.	Date of Election
Richard, G. A., Mount Morgan, Queensland	1895
Richards, A. H., Westport, N.Z.	1900
Richards, H. P., Mt. Boppy South, Boppy Mount, N.S.W.	F.
Ridge, H. M., c/o Institute of Mining and Metallurgy, Salisbury House, London, E.C., England	1910
‖Rigby, E. J., Austral Otis Engineering Co., South Melbourne, Vic.	F.
Roberts, J. Whear, Athenæum Club, Sydney, N.S.W.	1900
Rodda, E. J. J., Mungana (Chillagoe) Mining Co. Ltd., Mungana, Queensland	1907
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§Ross, C. N., Trinity College, Melbourne, Vic.	1907
Saunders, Geo., J., School of Mines, Charters Towers, Queensland	1904
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Schloesser, Robert, 60 Queen-street, Melbourne, Vic.	1905
Schmidt, W. H., Australian Metal Co., William-street, Melbourne, Vic.	1907
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‖Sarvaas, Johan, Bendigo, Vic.	1902
‡Scruby, H. M., Black Range, W.A.	1906
‖Seale, H. P., Mount Morgan, Queensland	1898
Simpson, Aug., Launceston, Tas.	F.
Simpson, W. Evan, Einasleigh Copper Mine, N. Queensland	1906
Skeats, Ernest W., University, Melbourne, Vic.	1905
§Skerritt, A. W., Chapple-street, Broken Hill, N.S.W.	1907
Slee, Richard T., Broken Hill Proprietary Mine, Broken Hill, N.S.W.	1902
Smith, A. Mica, School of Mines, Ballarat, Vic.	1906
Smith, Charles L., Hampden, Cloncurry, Queensland	1905
‖Smith, H. V., 54 Victoria-road, Hawthorn, Vic.	1901
Smith, R. Neil, Tasmania Club, Hobart, Tas.	1906
Snow, Francis H., National Mutual Buildings, King William-street, Adelaide, S.A.	F.
Somerset, H. St. J., Jun., Mount Morgan, Queensland	1898
‡Stanbury, T. R., P.O. Box 247, Kalgoorlie, W.A.	1902
‖Stewart, Hector, 12 Hornby-street, Windsor, Melbourne, Vic.	1902

Names and Addresses.	Date of Election
Sticht, Robert C., Mount Lyell Mining and Railway Company, Queenstown, Tas.	1901
||*Stirling, James, "Tormore," Highland-avenue, Hollywood, Los Angeles, Cal...	F.
Stirling, J., Burnie, Tas.	F.
Stoddart, Raymond, Sydney, N.S.W.	1907
§Strutt, S. S., Cassilis G.M. Co., Tongio West, Vic... ..	1906
Stubs, J. T., Lefroy, Tas.	1898
Süssmilch, C. A., Technical College, Sydney, N.S.W. ..	1904
Swanson, Fredk., Kalgurli G.M., Boulder, W.A.	1902
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Tennent, Bruce E., Witwatersrand Deep Ltd., P.O. Box 5, Knights, Transvaal	1904
||Thomas, A. C., Winmallee-road, Balwyn, Vic.	1904
Thow, Sydney, 55 Pitt-street, Sydney, N.S.W.	1902
Trude, F. B., Union Bank Buildings, Kalgoorlie, W.A. ..	1901
||Turnbull Thos. A., Richmond, Nelson, N.Z.	F.
||Turri, G. G., 364 Collins-street, Melbourne, Vic.	F.
Ulrich, George, Broken Hill, N.S.W.	1906
||Vale, W. H., Canterbury, Vic.	1894
Waddell, W. A., Koorboora, Queensland	F.
Wainwright, W. E., Broken Hill South S.M. Co., Broken Hill, N.S.W.	1902
Warde, Harvey S., Court House, Croydon, Queensland	1907
||Warren, John, Smith's Creek Tin Mines, Nymbool, N. Queensland	F.
||Waters, D. B., The University, Dunedin, N.Z.	1904
||Watt, Edwin O., Westralia Waihi Gold Mines N.L., Davyhurst, W.A.	1901
||Weinberg, E. A., 39 Queen-street, Melbourne, Vic. ..	F.
Weir, George, North Broken Hill Co., Broken Hill, N.S.W. ..	1902
Wentworth, D'Arcy, Mount Morgan, Queensland ..	1898
Wesley, W. H., Jun., Queenstown, Tas.	F.
Westcott, L. A., Many Peaks Mine, Nanandoo, Queensland	1907
White, Edmond A., Mount Morgan, Queensland	1900
||White, Norman F., Mount Morgan, Queensland	1899
§White, R. J., School of Mines, Bendigo, Vic.	1906
‡Whitehead, Edwin A., Box 59, P.O., Broken Hill, N.S.W. ..	1902

Names and Addresses.	Date of Election
Williams, Charles, Levin, N.Z.	1896
‖Williams, Luke, "Claremont," Moonah, Tas	1894
Williams, Richard, Broad Arrow Consols G.M. Co., Broad Arrow, W.A.	1902
Williams, Wm. G., St. Arnaud, Vic.	1907
Wilson, Gordon, Australian Gold Recovery Co., Charters Towers, Queensland	1901
Wilson, J. Bowie, Mount Morgan, Queensland	1898
Wilson, J. Kitchener, Moortgedacht Mine, Box 28, Lydenberg, Transvaal	1900
Woodman, Stanley S., Central Mine, Broken Hill, N.S.W.	1906
Wright, Geo. W., Queenstown, Tas.	1906
‖Wylie, Wm., 394 Great King-street, Dunedin, N.Z.	F.

LIST OF MEMBERS WHO HAVE HELD OFFICE.

	1893	1894	1895	1896	1897	1898	1899	1900	1901	1902	1903	1904	1905	1906	1907
Adams, R.	C	C	C	C	C	—	—	—	—	—	—	—	—	—	—
Ashcroft, E. A.	—	—	—	—	C	—	—	—	—	—	—	—	—	—	—
Ayers, Sir Henry, G.C.M.G.	P	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Blakemore, G. H.	—	—	—	—	C	C	C	C	C	C	C	—	—	—	—
Chapman, R. W.	—	—	—	—	—	—	—	—	—	C	C	C	C	VP	VP
Cleland, E. D.	T	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Cock, John	—	—	—	—	—	—	—	—	—	—	C	C	C	C	—
Courtney, C. F.	—	—	—	—	—	—	—	—	—	—	—	—	C	C	VP
Davey, T. G.	—	—	—	—	C	C	C	—	—	—	—	—	—	—	—
Deeble, M.	C	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Delprat, G. D.	—	—	—	—	—	—	—	—	—	—	C	C	VP	P	C
Dudley, Uriah	S	S	S	C	C	C	C	C	C	—	—	—	—	—	—
Gordon, Henry A.	—	—	—	—	—	—	—	—	—	—	P	—	—	—	—
Gray, W. B.	—	—	—	—	—	C	C	C	C	VP	VP	VP	—	—	—
Gregory, J. W.	—	—	—	—	—	—	—	—	—	C	C	C	C	—	—
Hamilton, R.	—	—	—	—	—	—	—	—	—	—	—	—	—	—	C
Hancock, H. Lipson	—	—	—	—	—	—	—	—	—	—	—	—	—	—	C
Hancock, H. R.	C	C	C	C	C	VP	VP	VP	—	—	—	—	—	—	—
Herman, H.	—	—	—	—	—	—	C	C	C	C	VP	VP	VP	C	C
Hodgkinson-Carrington, A. J.	C	C	C	C	C	—	—	—	—	—	—	—	—	—	—
Holcombe-Hewlett, E. F. J.	—	—	—	—	—	—	—	C	C	C	—	—	—	—	—
Holroyd, A. G.	—	—	—	—	C	C	C	—	—	—	—	—	—	—	—
Howell, John	VP	C	C	C	C	C	VP	VP	VP	C	—	—	—	—	—
Hunter, Stanley	—	—	—	—	—	—	—	T	T	T	T	T	T	T	C
Johnson, J. C. F.	C	C	—	—	—	—	—	—	—	—	—	—	—	—	—
Kayser, H. W. Ferd.	—	VP	VP	VP	VP	P	—	C	C	C	C	—	—	—	—
Kenyon, A. S.	—	—	—	—	S	S	S	S	S	S	S	S	S	S	C
Knapp, H. H.	—	—	—	C	C	C	C	C	C	—	—	—	—	—	—
Knox, Wm.	—	—	—	—	—	—	C	P	—	—	—	—	—	—	—
Koehler, W. J.	C	C	C	C	—	—	—	—	—	—	—	—	—	—	—
Lane, Z.	C	C	—	—	—	—	—	—	—	—	—	—	—	—	—
Lidgey, Ernest	—	—	—	—	—	T	T	C	P	C	C	C	—	—	—
McKay, J. R.	—	T	—	—	—	—	—	—	—	—	—	—	—	—	—
Merrin, A. H.	—	—	—	—	—	—	—	—	—	—	—	—	C	C	C
Montgomery, A.	—	—	P	P	C	C	C	C	VP	VP	VP	C	—	C	C
Morgan, P. G.	—	—	—	—	—	—	—	—	—	C	C	C	C	—	—
Moss, Frank A.	—	—	—	—	—	—	—	—	—	C	C	C	VP	VP	P
Park James	—	—	—	—	—	—	—	C	C	—	—	—	—	—	—
Pleitner, W.	C	C	—	—	—	—	—	—	—	—	—	—	—	—	—
Power, F. Danvers	VP	VP	C	C	P	C	C	C	C	C	C	P	C	C	C
Richard, G. A.	—	—	—	—	—	C	P	C	C	C	—	—	—	—	C
Rigby, E. J.	—	—	—	—	C	C	C	—	—	—	—	—	—	—	—
Rosales, H.	—	—	—	C	T	—	—	—	—	—	—	—	—	—	—
Schlapp, H. H.	C	C	C	T	VP	VP	VP	—	—	—	C	C	C	C	C
Sticht, Robert	—	—	—	—	—	—	—	—	—	—	—	C	P	C	C
Stirling, James	C	P	C	S	—	—	—	—	—	—	—	—	—	—	—
Stirling, J.	—	—	—	C	—	—	—	—	—	—	—	—	—	—	—
Süssmilch, C. A.	—	—	—	—	—	—	—	—	—	—	—	—	—	—	C
Tate, Prof.	VP	VP	VP	VP	—	—	—	—	—	—	—	—	—	—	—
Turri, G. G.	—	—	—	—	CT	C	C	VP	VP	VP	—	—	—	—	—
Wainwright, W. E.	—	—	—	—	—	—	—	—	—	—	—	—	—	—	C
Warren, John	C	C	VP	VP	VP	VP	—	—	—	P	C	C	C	C	C
Waters, D. B.	—	—	—	—	—	—	—	—	—	—	—	—	—	C	C
White, W.	C	C	C	—	—	—	—	—	—	—	—	—	—	—	—

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PAPERS AND DISCUSSIONS.

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NOTE.—This paper is issued for discussion by members. Correspondence is invited. Discussions will be published in a subsequent number of the Proceedings, and, when finally closed, the paper, with discussion in full, will be published in the Transactions of the Institute.

This paper is separately bound, and may be, if so desired, detached complete from this number.

BROMO-CYANIDING OF GOLD ORES.

BY E. W. NARDIN.

ALTHOUGH the bromo-cyanide or Diehl process for the extraction of gold from its ores is used on several of the Kalgoorlie mines, very little information has been published on the original process, and no details are available from the various mines now using it.

The following description of the process as carried on at the Hannan's Star plant, where several modifications were introduced by the author, will therefore be of interest to mining men, and it is hoped that some discussion on this paper will provide further useful information to the mining community.

The process was first installed on the Hannan's Star mine, where a full working plant was erected by the London-Hamburg Co. In their contract they guaranteed a 91 per cent. extraction on a sulphide ore of 15 dwt. value. This was considered a very high extraction at the time, and the plant successfully carried out its work.

The new features in the process were the finer grinding of the ore, for which purpose the tube mill was introduced, and the addition of a solution of bromo-cyanide to the vats under ordinary cyanide treatment, whereby a higher extraction was obtained in a shorter time. The main difficulty in the process was the easy manufacture of the bromo-cyanide solution, but this was eventually overcome. The mixed salts now supplied by the London-Hamburg Co. are in every way satisfactory, and are supplied at such a price that the process compares favourably with any other from a cost point of view.

Most of the Hannan's Star ore which has been treated by the process was of low grade, and hence the percentage extraction shows low, but the average value of residues for 1904 and 1905 were 1 dwt. 2 grs. and 1 dwt. respectively on ore averaging 10 dwts. 7 grs. and 8 dwt. 9 grs. respectively, which must be considered good work. On high grade ore the process was equally successful. Some 15,000 tons from the Brown Hill Extended mine, averaging about 4 ozs. gold per ton, and containing telluride, were treated by Mr. O. B. Ward in 1903 for an average extraction of 95 per cent. On Hannan's Star ore the average cost for bromo-cyanide, extending over a period of twelve months (September, 1904, to October, 1905), was 9.75 pence per ton of ore treated, and the extraction was entirely satisfactory. On the same class of ore (sulphide), roasting would cost about 2s. 9d. per ton, so that we have a balance of 2s. per ton in favour of the chemical, and the saving of very heavy first cost in furnace plant.

In other places, where fuel and furnace supplies were expensive, the chemical process would show a much more decided advantage.

The process requires more metallurgical skill and constant attention to the progress of each vat under treatment; but, if this is available, the results are highly satisfactory, and the process has very definite claims as a cheap and efficient method of treatment for such ores as the Kalgoorlie sulpho-tellurides.

The author took charge of the Hannan's Star plant in July, 1904, and the process at that time was as follows:—

The ore from the rock-breaker was dry crushed in two No. 5 Krupp ball mills, with steel wire screens of 27 holes per lin. inch. From the mills it passed to a mixer, over amalgamated plates, and then to a classifier at the end of the tube mill. Here a separation was made, the overflow going to the slimes pumps, and the underflow into the tube mill. The discharge from the mill was elevated back to the classifier, where separation of the insufficiently ground portion was again effected. The slimes pumps delivered to two sets of spitz-luten, the underflows from which passed back to the tube mill, and the overflows to the pointed settling boxes, where the pulp was thickened up before passing to the agitator vats. The point aimed at was to slime the whole of the ore to pass a sieve of 150 holes per lin. inch. In practice about 5 per cent. would remain on the "150" sieve, but the whole of it would pass the "100" sieve.

There were four agitator vats of 50 tons dry ore capacity each. These each took on an average 16 hours to fill, representing 75 tons per day through the ball mills. Their tonnage was determined, for working purposes, by weighing the contents of a bottle, filled to its containing mark by dipping into the vat, and by measuring the volume of pulp in the vat. From the volume, tonnage, and specific gravity, the dry weight of ore was obtained.

A mechanical sampler was arranged between the ball mills and the mixer, which cut out a definite quantity periodically. This bulk sample was "quartered" down, which gave the ore sample for the day. A vat when full was given its charge of KCy, and three hours afterwards a "dip" KCy residue was taken, and the charge of BrCy solution added. After a total agitation of 20 hours a quantity of lime was added and the vat "pressed." The quantity of BrCy added was varied according to the residues of preceding vats and the value of the ore being treated as shown by the daily ore sample.

Every sixth frame in a filter-press was sampled, and the bulk sample from the whole vat quartered down to represent the final residue of that vat. The KCy residue was washed on a small vacuum filter and assayed, while the final residues were assayed after drying. From the daily ore samples KCy and final residues, the total extraction and extraction by plain KCy and by BrCy were determined.

Bromo-Cyanide Solution.—The bromo-cyanide solution is made according to the following equation :—

$$(1.)\ 2\,KBr + KBrO_3 + 3\,KCy + 3\,H_2SO_4 = 3\,BrCy + 3\,K_2SO_4 + 3\,H_2O.$$

And its action in the treatment vat is supposed to be as follows :—

$$(2.)\ BrCy + 3\,KCy + 2\,Au = 2\,K\,AuCy_2 + KBr.$$

The first two quantities in equation 1 are contained in the mixed salts supplied by the London-Hamburg Co., and which have about the following composition :—

40–44 % of *Br* as KBr
20–22 % of *Br* as $KBrO_3$;

the proportion of *Br* as bromide being about twice that of *Br* as bromate. A 30-lb. charge is usually made up, and for this the following weights are taken :—

H_2SO_4	50 lbs.
KCy	20 lbs.
Mixed salts	36.8 lbs.

The KCy being 93 per cent., and the H_2SO_4 63 per cent. (chamber acid). The solution is made in a closed wooden vessel, stirred by rotating arms, and holding about 200 gallons. In making up a charge, portion of the water and all the H_2SO_4 are first mixed, and allowed to cool to normal temperature. The KCy, which is dissolved in a separate vessel in sufficient water to fill the mixing vessel, is then run in, and at the same time the proper weight of "mixed salts" gradually added. The whole is then agitated for six hours before being used, and in a closed vessel will retain its strength for some days.

The cost of a 30-lb. charge of BrCy is about £4 10s., made up as follows :—

50 lbs. H_2SO_4 at 2d., 20 lbs. KCy at 8.37d., 36.8 lbs. salts at 1s. 10d.

From each charge mixed, a dip sample is taken and tested with a standard $Na_2S_2O_3$ solution, using potassium iodide as an indicator.

$$\text{(3.)}\ \underset{(106)}{BrCy} + 2\,KI = KBr + KCy + \underset{(254)}{I_2}$$

$$\text{(4.)}\ \underset{(316)}{2\,Na_2S_2O_3} + \underset{(254)}{2\,I} = 2\,NaI + Na_2S_4O_6$$

Thus BrCy corresponds to 2 $Na_2S_2O_3$.

The standard solution is made so that 1 c.c. = .02 BrCy, and for this about 93.6 grams of ordinary photographic crystals, $Na_2S_2O_3\ 5\ H_2O$, are dissolved in 1 litre of water.
(248)

A solution of copper sulphate is used for standardizing the above.

$$\text{(5.)}\ \underset{(126.8)}{2CuSO_4} + 4\,KI = Cu_2I_2 + 2\,K_2SO_4 + \underset{(254)}{I_2}$$

126.8 Cu liberates 254 I, so that it corresponds to 106 BrCy and 496 $Na_2S_2O_3\ 5\ H_2O$.

The solution is made by dissolving one gram of pure copper foil in acid, converting to sulphate, and dissolving in 100 c.c. water.

10 c.c. contains .1 grain Cu = .0836 BrCy;
then 1 c.c. hypo solution = ·02 gram BrCy.,
and 4.18 hypo solution = ·0836 gram BrCy.

In testing BrCy solutions, 5 c.c. are usually taken, Na_2CO_3 solution added till alkaline, and then acetic acid till acid. A few crystals of KI and some starch solution are then added, and the whole titrated with the standard $Na_2S_2O_3$ solution.

Example :—If 5 c.c. BrCy solution took 3.2 c.c. hypo, then

$$\frac{3.2 \times .0836}{4{\cdot}18} \times 20 = 1.28\ \%$$

This method of testing BrCy solutions is different to that used on other mines, where they titrate direct with $Na_2S_2O_3$ without first neutralizing the H_2SO_4. The following is a series of tests made on a number of 30-lb. BrCy solutions by direct method, A, and, after neutralizing, B :—

	A.	B.
Charge 668—30 lbs.	·36 %	1·20 %
Charge 670—30 lbs.	·44 %	1·20 %
Charge 671—30 lbs.	·60 %	1·28 %
Charge 672—30 lbs.	1·20 %	1·24 %
Charge 673—30 lbs.	·36 %	1·36 %
Charge 674—30 lbs.	·52 %	1·00 %

These show that by direct titration the test is usually low. and also that it gives irregular results, depending on the amount of free H_2SO_4 present. There seems to have been some doubt as to whether a BrCy solution increases in strength after, say, one hour's agitation, but the following tests, which were made, show that it does increase up to about 8 hours.

After	1	hour	mixing solution	tested	0·56 %	BrCy.
,,	2	,,	,,	,,	0·72	,,
,,	3	,,	,,	,,	0·72	,,
,,	4	,,	,,	,,	0·80	,,
,,	5	,,	,,	,,	0·80	,,
,,	6	,,	,,	,,	0·88	,,
,,	7	,,	,,	,,	0·92	,,
,,	8	,,	,,	,,	1·00	,,
,,	24	,,	,,	,,	1·04	,,

At the time the author took charge it was generally known that if "plant" solution became too alkaline, either through a change in the ores, or the addition of too much lime either to the ore before crushing or after the bromo-cyanide treatment, that the extraction by BrCy fell considerably, and for this reason an occasional test of the plant solution was made for alkalinity, but not till several high residues had occurred.

Under the old system, too, the KCy did not have sufficient time by itself; the values of the ore and KCy residues of each vat were not known, and the amount of BrCy that should be added was more or less a guess. When it is remembered that every 5 lbs. of BrCy added to a 50-ton vat represents a cost of 4d. per ton of ore, that the action of the BrCy can be made just as effective after long KCy treatment, and that any excess of BrCy is of no advantage, it will be seen how important it is that the value of the KCy residue should be known after sufficient agitation (say 12 hours), and the condition of the vat tested as to alkalinity before adding the BrCy. In any case the action of the BrCy is of short duration, not exceeding 4 hours, so that if 20 hours' total agitation can be allowed, it is better to give 16 hours with KCy and then add the BrCy. It is preferable, however, to keep the vat under KCy treatment until the KCy residue is known, then correct the alkalinity and add the BrCy. This could easily be done with extra vat capacity. Total alkalinities are determined with standard solutions of HCl and NaHO, using phenolthalien as indicator. The following experiments were carried out to determine what was the most suitable degree of alkalinity for bromo-cyaniding.

Six sludge samples were taken at the same time from a vat which had had KCy treatment. The solution was filtered off one, and the quantity of a weak H_2SO_4 solution required to about neutralize it was determined. It took 40 c.c., so that to the sludge samples was added the following quantities of H_2SO_4 solution respectively :—

No. 1 ..	.. 40 c.c.
No. 2 ..	.. 35 c.c.
No. 3 ..	.. 26 c.c.
No. 4 ..	.. 21 c.c.
No. 5 ..	.. 14 c.c.
No. 6 ..	.. 7 c.c.
No. 7 ..	.. *Nil*

To each sample was also added 7 c.c. of BrCy solution, and the bottles were sealed and agitated for 8 hours. The solutions were then poured off and tested for alkalinity, while the residues were washed and assayed. The following were the results :—

	Alkalinity.	Value of Residue. dwt.	grs.
No. 1	·0096 %	0	21
No. 2	·0128 %	0	21
No. 3	·0172 %	1	0
No. 4	·0262 %	1	0
No. 5	·0377 %	Lost	
No. 6	·0595 %	1	3
No. 7	.0800 %	1	9

The KCy residue of the vat was 2 dwts., the final "press" residue 1 dwt. 9 grs., and the alkalinity of the solution from the KCy residue was ·08 per cent., and these are included as No. 7 in the above table. From these tests it will be seen that as the alkalinity was reduced so the residues fell in value in almost exact proportion; they prove conclusively that BrCy does not act well in a too alkaline solution, and that the best action is obtained when the alkalinity is about ·01 per cent. to nearly neutral.

A second series of tests on a different value ore (about 14 dwts.) was made; as in the previous case varying quantities of H_2SO_4 and 7 c.c. of BrCy were added to the bottles, agitated for 12 hours, and the residues washed and assayed.

	Alkalinity.	Value of Residue. dwt.	grs.
No. 1.—*Nil*	·0134	1	9
No. 2.—3 c.c. acid	·0083	1	6
No. 3.—6 c.c. acid	·0064	1	0
No. 4.—8 c.c. acid	Nearly neutral	1	0

The KCy residue of this vat was 4 dwts. 12 grs., and the final press residue 1 dwt. 9 grs. It became the practice, therefore, to have the alkalinity of the vats during KCy treatment at between ·02 and ·03 per cent., to keep the vat agitated till the value of the KCy residue was known, then correct the alkalinity to about ·01 per cent. or less by H_2SO_4 and add BrCy according to the value of the KCy residue. A better extraction was thus obtained at less cost for BrCy, at the expense of a small amount of H_2SO_4 and extra agitation.

Owing to an insufficient number of vats, they could not always be kept back for the KCy residue, but on every occasion on which it was done a certainty could be made of the residue being low and the consumption of BrCy a minimum. A number of experiments were carried out to explain this reduced action in an alkaline solution, but nothing definite was proved, although it is possibly due to the presence of KHO.

If a solution of KHO be added to BrCy it is almost immediately destroyed. No smell of BrCy is left, and no test can be obtained with $Na_2S_2O_3$.

$$\text{(6.)}\quad BrCy + 2\,KHO = KBr + KCyO + H_2O.$$

Action of BrCy on FeS_2 and metallic Fe.—Various experiments were carried out to determine the action of BrCy on FeS_2 and finely divided metallic Fe.

The FeS_2 used was panned off from finely crushed ore, while metallic Fe was picked by a magnet from the sludge in a vat. Freshly prepared iron filings were also tried. Small quantities of these were agitated with a previously tested BrCy solution. Action at once began, and in a short time the BrCy was destroyed. All smell of BrCy disappeared, and no test could be obtained with $Na_2S_2O_3$. The result is a greenish-blue solution of a ferrous salt, from which a dark green ferrous salt is precipitated by NH_4OH.

These experiments showed that both FeS_2 and metallic Fe are bromo-cyanicides, that finely-divided metallic Fe is the more active, and that there is sufficient of the latter produced in grinding the ore to destroy the amount of BrCy usually added to a vat.

The BrCy seems to act more by breaking up the FeS_2 than by actual solution of the gold, but it is probable that a great portion of it is destroyed by the finely-divided metallic Fe, without, of course, any beneficial result as far as setting free or dissolving gold is concerned.

Effect of adding Lime.—Experiments were carried out to show the effect of adding ordinary commercial lime to vats after BrCy treatment, and it was found that a re-precipitation of gold took place to a small extent. To show this in a magnified way, two samples of solutions—A and B—were taken from a vat after BrCy treatment. An excess of lime was added to A, agitated for a few hours, and then assayed, with the following results :—

A solution assayed 3 dwts. 21 grs. per ton ;
B solution assayed 5 dwts. 5 grs. per ton ;

showing that the lime precipitated 1 dwt. 8 grs. of gold per ton from solution, or 25 per cent. of its contents.

This is probably due to the presence of carbon or occluded gases, such as hydrogen.

The results of these experiments have suggested certain improvements in bromo-cyaniding, some of which the author was able to adopt, and the others are recommended. They may be summed up as follows :—

1. The daily ore sample should be taken in the morning, and assayed as soon as possible, so as to know what was the value of the ore passing to the vats in the previous 24 hours.
2. The pulp should have a long KCy treatment.
3. A vat should be kept under KCy treatment till the value of the KCy residue is known.
4. The alkalinity of the vat should then be determined and corrected to ·01 per cent. by H_2SO_4 before adding BrCy.
5. The quantity of BrCy added should then be determined from the value of the KCy residue, the tonnage of the vat, &c.
6. The lime added to the ore during crushing should be varied according to the alkalinity test after KCy treatment, so that the plant solution tests about ·02 per cent.
7. Lime water should be made and added to the vats or to the solution from the presses, instead of adding lime to the vats.
8. All metallic iron should be removed in some way from the ore before treatment, as it is both a cyanicide and bromo-cyanicide.

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*THE REFINING OF BASE BULLION AT PORT PIRIE, AND TREATMENT OF BY-PRODUCTS.

By Brian B. Bayly.

Preface.

The article, as written, represents a short account of the treatment of the base bullion from the Broken Hill ores, as carried on at Port Pirie, for the recovery of lead, silver, and gold, and for the treatment of by-products. Appended is a short description of the chief facilities of plant which have been adopted to overcome the main difficulties in the refining of lead—the Howard press and stirrer. The article is within the limit prescribed by the regulation for the prize offered by the Australasian Institute of Mining Engineers, comprising 30 pages, containing an average of about 200 words per page, exclusive of analyses, photographs, and sketches.

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* Student's prize paper.

The writer desires to thank those to whom he is indebted for their kindness in affording him every facility possible to obtain the information which he required for the success of such a paper—viz., Messrs. Robertson, Leavens, and Gronow, of the Broken Hill Proprietary Co. Limited, at Port Pirie, and Mr. Crowther, of the Adelaide School of Mines, to whom he is indebted for photographs.

The difficulty of obtaining fair samples of gold drosses, and the inability to obtain particulars of fuel consumptions, prevents the paper from being as complete as one would wish. The limited length precludes detail, except in more important parts.

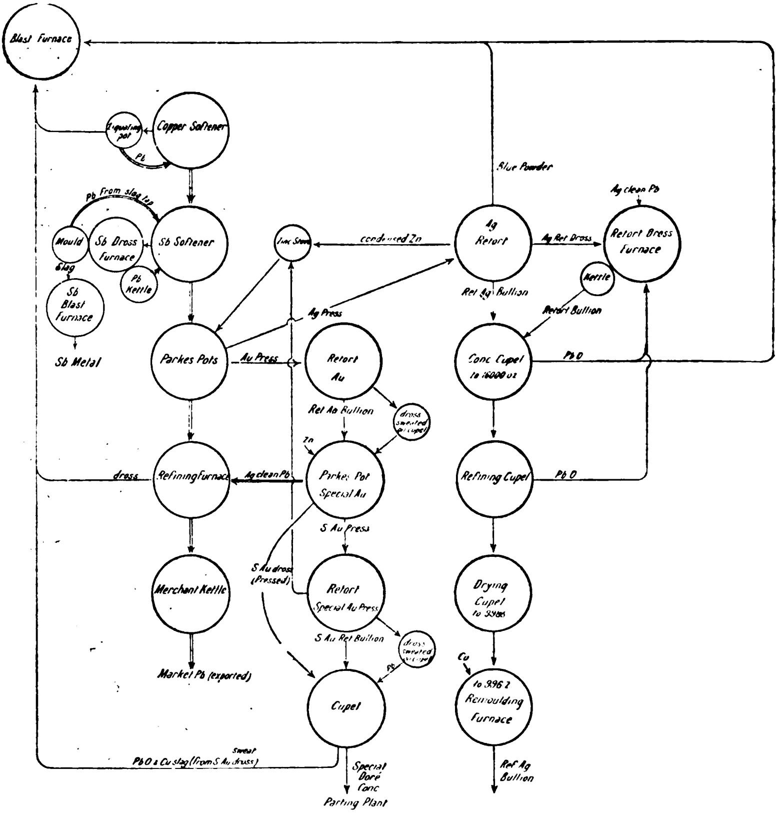
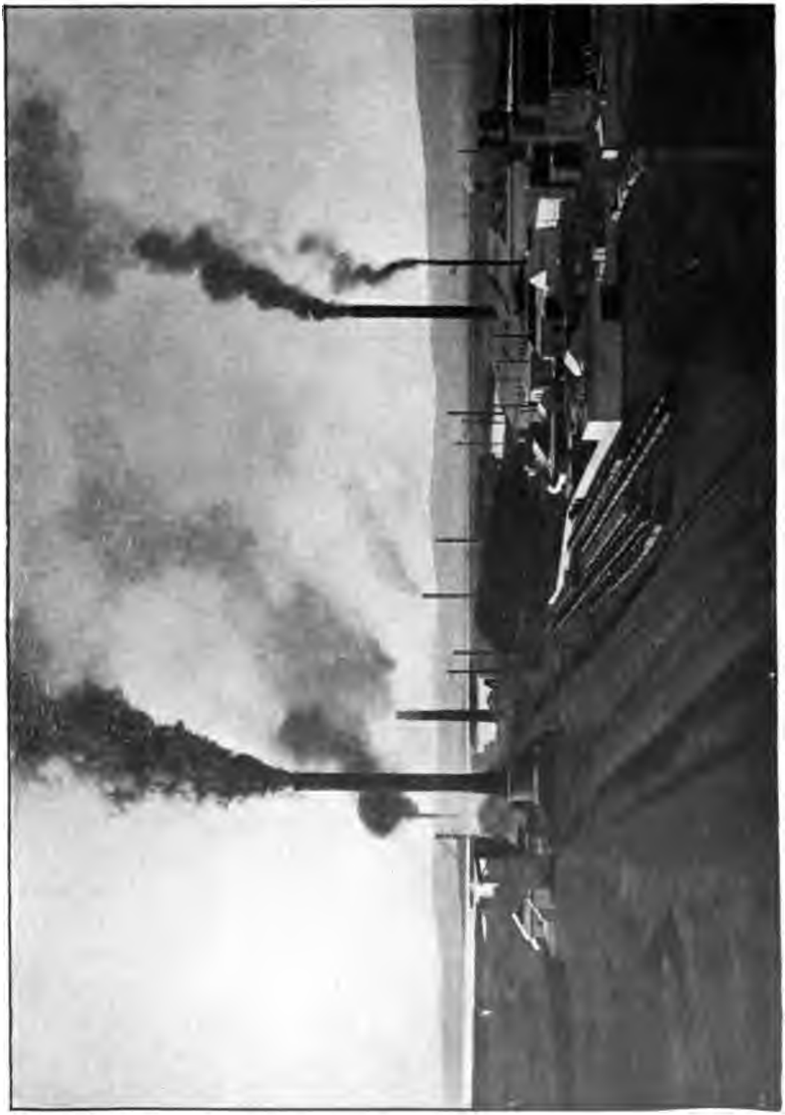


FIG. 1.—DIAGRAM OF RECOVERY OF LEAD AND SILVER AT PORT PIRIE.

BAYLY
ON REFINING BASE BULLION.
PLATE I.



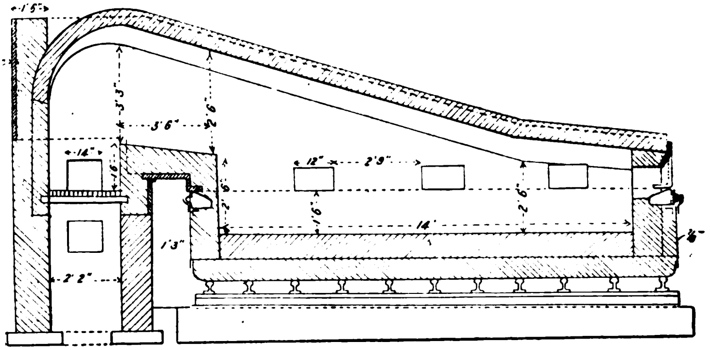
SMELTING WORKS, PORT PIRIE—GENERAL VIEW.

The base bullion is brought in carts by contract to the refinery from the blast furnaces, and dumped opposite the copper softener. The bullion bars, weighing 80 to 90 lbs. each, are run into the copper softener by means of a peel, similar to the baker's implement.

The following is an analysis of base bullion from the smelters :—

Insoluble	·088 per cent.
Cu	·452 per cent.
As	·124 per cent.
Sb	·201 per cent.
Fe	·039 per cent.
Zn	·049 per cent.
S	·177 per cent.
Pb	98.87 per cent.

The furnace is of the reverberatory type, built of Gartcraig fire-brick and black New Zealand fire-clay. The whole hearth is cased in $\frac{7}{8}$-in. iron plate tank, in case of collapse of the fire-bricks, when the whole charge may be caught. There are three charging doors on one side, and one skimming door at the end of the furnace, and above the latter is a horizontal flue, connected with the underground refinery flue and stack. It is fitted with a hand-hole for cleaning (see Copper Softener, Plate III.). There is a water-jacketed tap-hole, closed by a wooden plug, leading down to the antimony softener, and round the furnace at lead level runs a water-jacket 8 in. wide, fed in two sections, which the corrosive action of the litharge on fire-brick requires (fig. 3). Beneath this water-jacket, at the junction with the further wall, is a weak spot, and to protect this an apron is provided which overlaps the jacket on the outside. The joint is rammed tight with cupel material. The roof of the hearth is arched both longitudinally and laterally; the bed is also an inverted arch in cross section (fig. 2), sloping to the tap-hole. The iron tank rests upon two rows of railway iron at right angles, which in turn rest upon a layer of bricks, which again rest upon a foundation of slag.



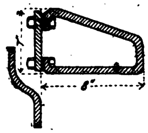


FIG. 3.

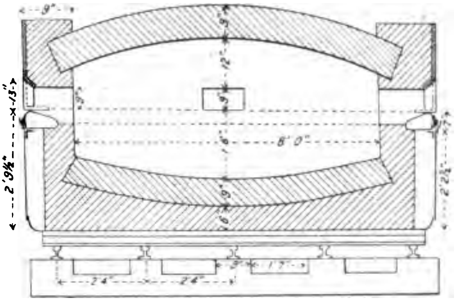


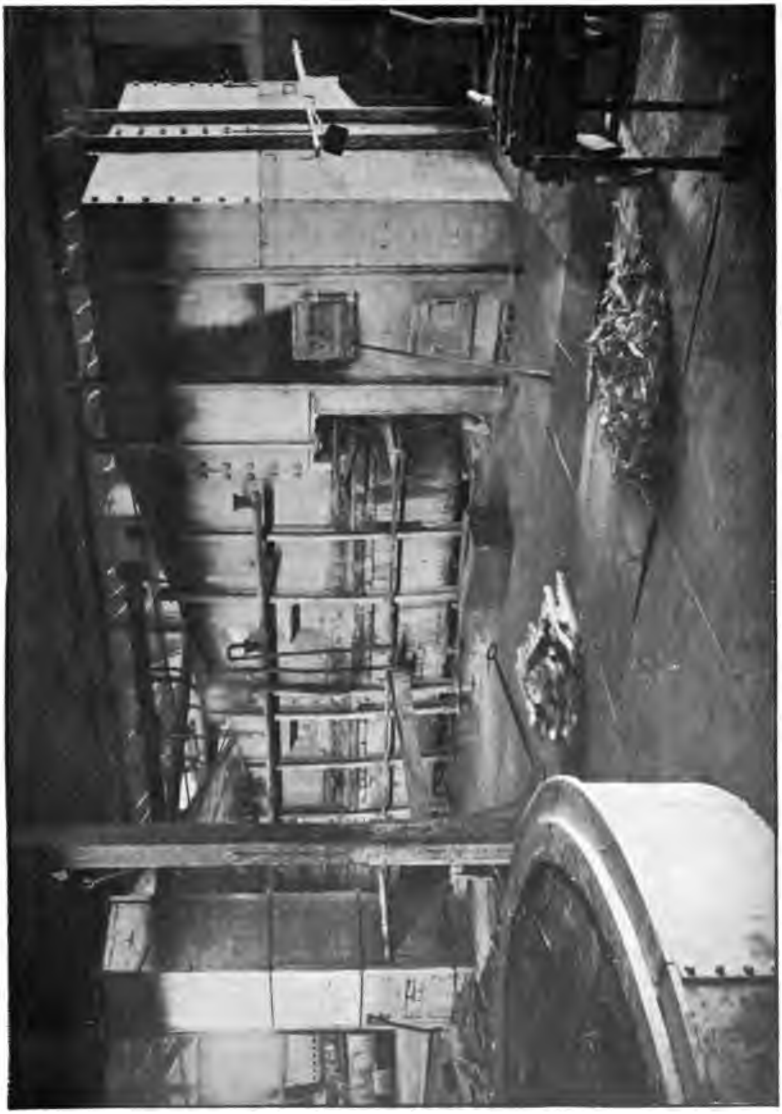
FIG. 2.

SKETCHES SHOWING DETAILS AND DIMENSIONS OF COPPER SOFTENER.

The sides and ends of the furnace are securely braced by buckstaves, united by tie rods at top and bottom, both length and crosswise. At the crown of the arch there are seven draught holes 4 in. x 4 in., which serve the purpose of cooling and preserving the roof. When new, the furnace will hold about 43 tons, but through wear it becomes somewhat enlarged. It requires chipping about once a month, owing to the formation of accretions in the hearth. These chippings go back to the smelters.

Between 1,000 and 1,100 bullion bars from the smelters form a charge for a copper softener, and charging takes four hours. The fire, previously kept low, is got fairly under way and regulated at a

BAYLY
ON REFINING BASE BULLION.
PLATE II.



ANTIMONY SOFTENER

moderate temperature to avoid melting the copper dross which rises to the surface of the metal, and which in the molten state would mix into the lead and spoil the silver. After heating for about three hours, the scum is skimmed off by means of an iron skimmer (fig. 4) into liquating pots—ordinary slag pots with a tap-hole near

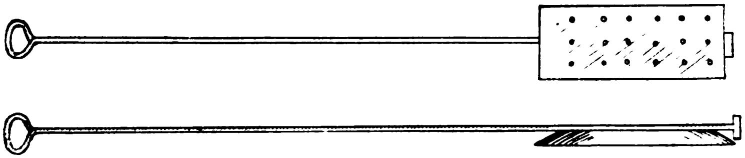


FIG. 4.
SKIMMER.

the bottom through which, by removal of the wooden plug, any contained lead may be drained off when the pot is full of dross.

Analysis of Copper Dross.

Pb	77·4 per cent.
Ag	31·6 ozs.
Cu	8.52 per cent.
Au	·3 ozs.
Fe	·73 per cent.
Zn	·86 per cent.
S	6·24 per cent.
As	·39 per cent.
Sb	·53 per cent.

Bullion from Copper Softener Impurities.

Cu	·125 per cent.
As	·094 per cent.
Sb	·18 per cent.
	———
Ag ..	70–80 ozs. per ton
Au ..	1 dwt.

In this way two tons of copper dross are taken from the furnace and re-treated in the smelters; the liquated lead is put back into the furnace, and, as soon as the antimony softener is free from a previous charge, the product of the copper softener is run in by a whitewashed gutter of channel iron. Both at the copper softener and here two men are required to work the furnace—one skimmer and a man to work the scum forward to the door.

The method of separating arsenic and antimony from the lead is to heat strongly enough in an oxidizing atmosphere to form a layer of litharge on top which absorbs these impurities as arseniates and antimoniates. The higher temperature required necessitates more fuel and stronger bracings for the bed. The details of doors, water-jacketing, accessories, and dimensions are identical with those given for the copper softener (Plate III.). The channel iron is supported by wooden trestles and the joints are made tight with clay. The charge is then run down in molten condition, and, together with the bullion from the antimony dross furnace, forms the complete charge for the second softening.

Skimming Composition from Antimony Softener.		Bullion from Antimony Softener contains impurities to the extent of	
Pb	73·88 per cent.	Cu	·12 per cent.
As	1·34 per cent.	As	*nil*
Sb	8.02 per cent.	Sb	·089 per cent.

Au and Ag same as Cu softener practically.

It is then fired hard for six hours, and if it contains much antimony will need to be poled with iron rakes to bring all the antimony into contact with the litharge. The rakes are worked down the sides and up in the centre. It is then left to cool back until the litharge crust has solidified, which is skimmed off and taken to the antimony dross furnace for treatment. Fire clean and fire upon the charge for four hours hard. Take a sample of the litharge and allow the furnace to "go back" by raking out most of the fire and opening the working doors.

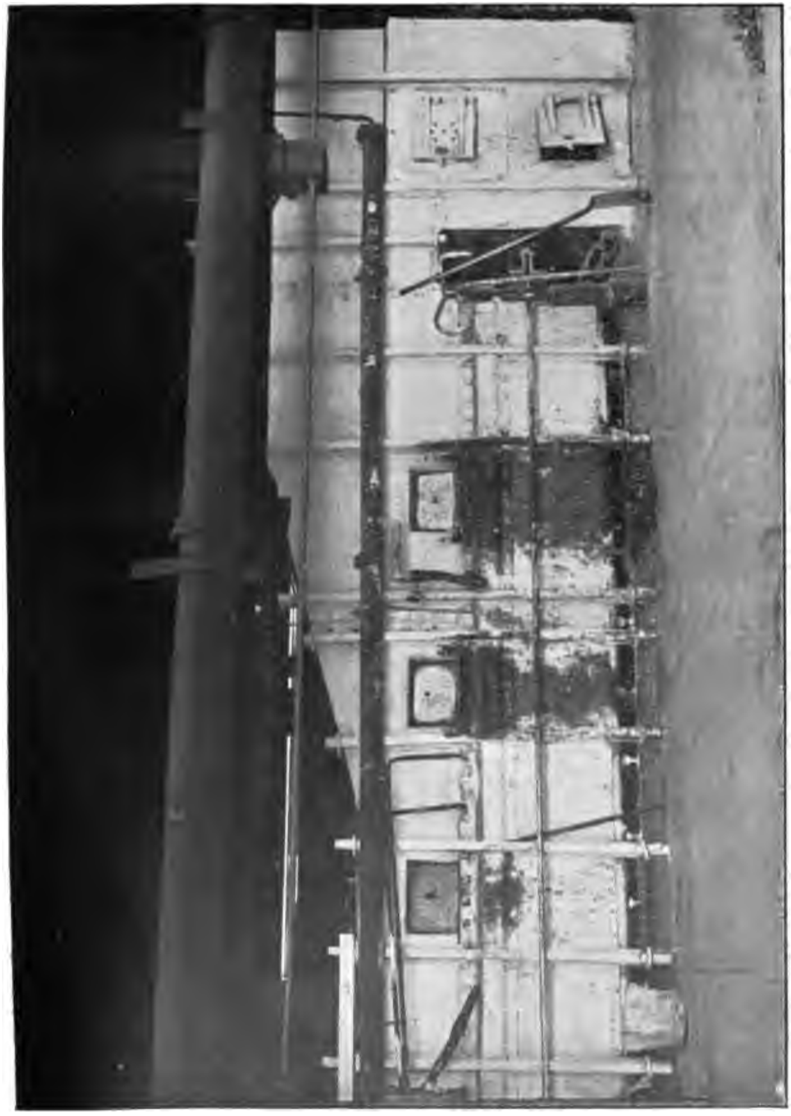
If when cold the litharge is highly crystalline and yellow, it will not be necessary to completely skim again; that nearest the skimming door only is taken off, and the rest left in the furnace for the next charge. At times, in spite of the litharge test, the lead still contains too much antimony, and when this is run into the Parke's pots it may need to be poled with wood and skimmed; but, if the sample be black and not so crystalline, there is too much antimony still, and another skimming is necessary.

The life of the furnace is about four weeks, when the bricks which have been slagged (mostly at the litharge level) must be renewed.

The weight of skimming is about one ton, and will be dealt with under "Treatment of By-Products."

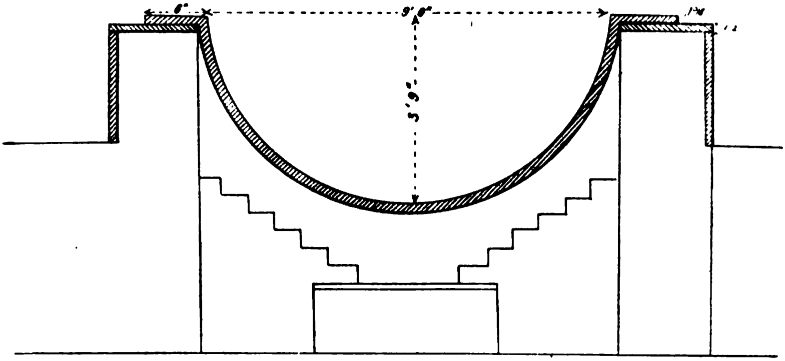
PROCEEDINGS AUS. I.M.E, VOL. III., NO. II.

BAYLY
ON REFINING BASE BULLION.
PLATE III.



COPPER SOFTENER

The zincking kettles, before receiving their charge, must be washed and heated up by means of fires underneath; the lead must also be cooled back in the antimony softener, so that, when run in, there is no appreciable difference of temperature between the two, which would cause the pan to strain, and perhaps crack. The kettle itself is a circular cast-iron pan of 9 ft. diameter and 3 ft. 9 in. depth.



Section of Zn pan with dimensions

FIG. 5.

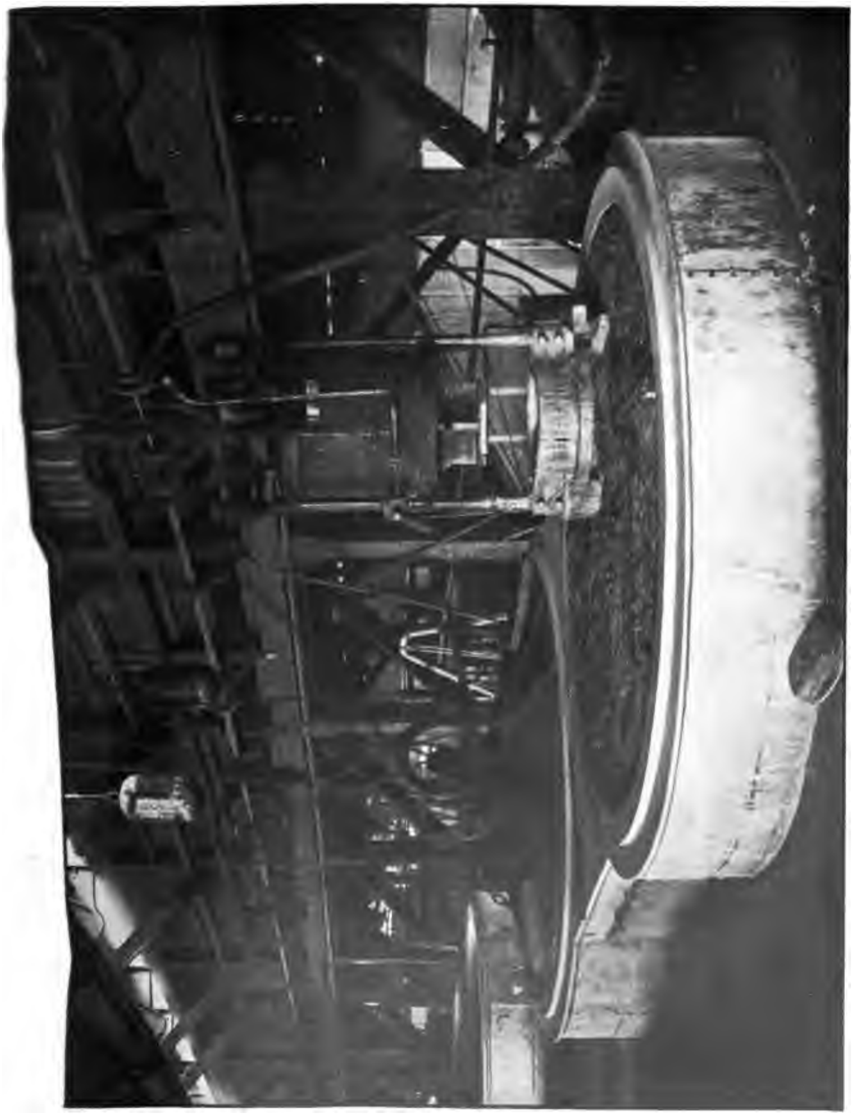
This will hold, approximately, 45 tons of metal. The 6-in. flange of the kettle forms the support on which it rests, being supported above the fire-box by a circular brickwork structure capped with 1½-in. plate of cast-iron, and on this rests the shoulder of the kettle. The firing is done from the next floor, grate area being 2 ft. x 8 ft. A step hearth is used, so that the flames must go within a foot of all parts of the kettle when the hearth is clean. As the working goes on, however, the hearth gets choked up with ash till this space is greatly diminished. If the space is not uniform all round the kettle this will produce unequal heating, and severe strains in the iron.

The principle upon which the separation of gold, silver, and lead is based is that these first two metals form an alloy with zinc more readily than with lead; this alloy is both more infusible and lighter than lead, consequently it rises as a scum, which may be removed on cooling the pot back. The gold and any copper present have a greater affinity for zinc than silver has, and may be separated before the lead is saturated, whereas silver does not alloy until after the saturation point of lead by zinc is reached. This forms a ready means of separation of gold and silver.

In former times these skimmings were treated first in a liquating furnace, and, after liquating the lead from the zinc, were treated in cupels for recovery of gold and silver; but in this way such a lot of lead was lost as litharge, in getting the gold and silver, that the skimmings were again put back into a pan richer than that from which they came, to absorb more precious metal, being finally pressed off in a Howard press.

Considering the "gold skims" first: the pans are worked in pairs. The zinc is added in several small additions, thus giving a more perfect separation of gold and silver, and a more efficient use of zinc than one large addition. The gold is practically all taken out before the silver by returning the skims from one kettle to another which is richer in gold. In this way, zinc, which is added, passes through three or perhaps four different charges for gold, being finally pressed off as a "first gold press." Then 80–100 lbs. zinc is added to this kettle, and a "second first" is skimmed off, which in its turn is used again. Thus the clean zinc is only added to a charge after the "first" has been taken off. Starting with pure zinc and no returns: the first zinc has to be returned, and returned alternately through the two pots until the fifth time through, when it is pressed as a "first." To clean the first charge in A of gold (fig. 6), 80–100 lbs. zinc is added, skimmed, and stacked ready to pass through B as "second first," whence it is skimmed and stacked as a "second times through." To clean B of gold 80–100 lbs. zinc is added, and the skimming with 150 lbs. zinc passes through the third charge, which is in pot A—the first charge having been desilverized and run down into the refining furnace—and again through a fresh charge in B, and finally stacked with the "second times through." Then both the "second times through" are together put into the fifth charge in A and pressed off as "the first gold press." The second firsts from the third, fourth, and fifth charges pass into the sixth charge in B, and are stored as before.

BAYLY
ON REFINING BASE BULLION.
PLATE IV.



ZINCKING PANS AND HOWARD PRESS.

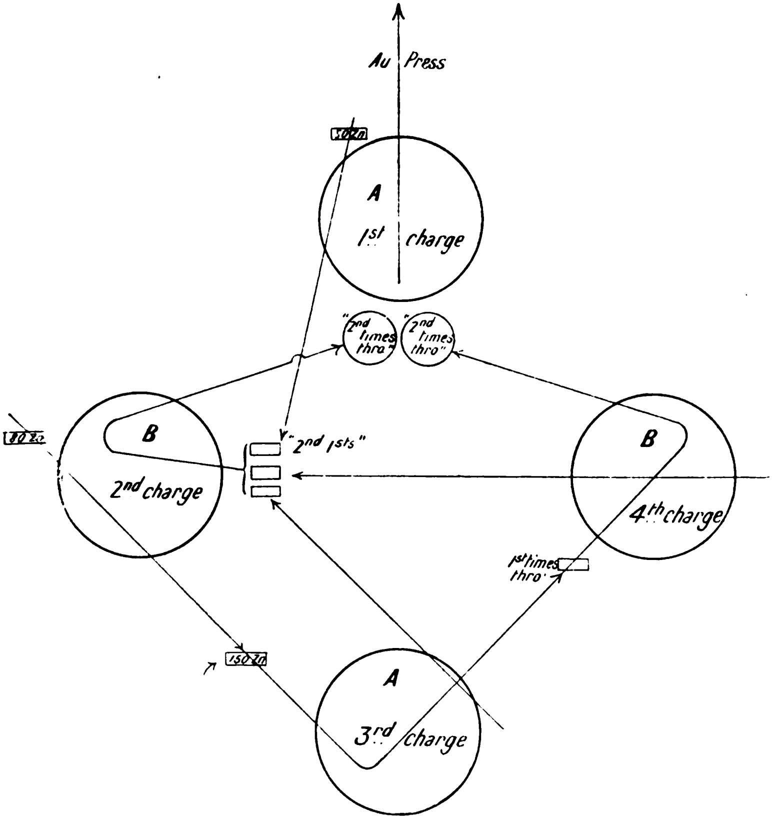


FIG. 6.

DIAGRAM REPRESENTING THE GOLD ZINCKING SCHEME.

WORKING WITH A PAIR OF KETTLES.

(From a Scheme drawn up by Mr. Crowther.)

The sixth charge in B is cleaned of gold as before, and the process goes on in a cycle.

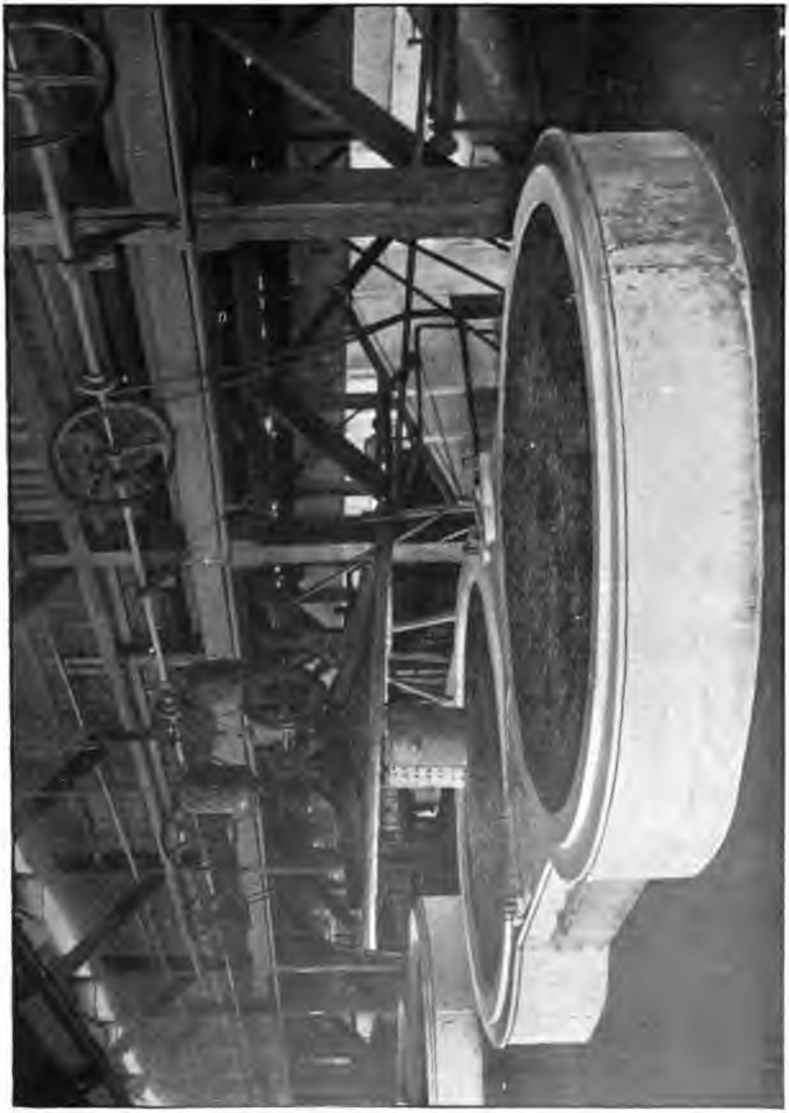
The Silver Skims Form Another Cycle.—A quantity of zinc calculated from assay of the silver in the lead charge is added—about 500 lbs. spelter—skimmed and pressed, forming the main silver press—" the second skim." Then an additional 734 lbs. is added to clean the lead of the last traces of silver, and is skimmed as "the third skim." This forms the "second skim" for the alternate kettle. In the cycle fresh zinc is only added for the "third skim," passes through the other kettle as "second skim," and is then pressed off. If the lead now goes more than ·3–·4 ozs. silver, another zincking is given, and is skimmed very cold.

The time of operation from charge to discharge in the zinc pans is about 32 hours, being roughly divided in two gold skims of eight hours each, and two silver skims of eight hours. The gold zinckings are heated up 1½ hours, stirred ½ hour, cooled back 4½ hours, and skimmed ½ hour, being left from an hour to 1½ hours for a sample to be taken. The silver zinckings are given more time for heating and cooling than this. The weight of skimmings taken from the kettles is about six tons. Two men are employed on each pair of pans—one right-handed and one left-handed skimmer. The life of the pans is very variable; they may crack after two or three charges. The lead is removed if possible before the pan collapses, and the pan is taken out and broken up, the scrap iron being used in fresh castings.

The Gold Pressings.—The first gold press, after being broken on the floor, is taken to the gold retorts and retorted for four hours, no attempt being made to condense any zinc, of which there is comparatively little. The copper has the effect of concentrating the gold irregularly in the bullion, making it almost impossible to get a fair sample and assay of it. Very roughly it would go about 12 ozs. to 1,000 ozs. doré bullion. A dross is skimmed from the retort; this is sweated on a bath of lead in a cupel. The gold retort bullion is stacked, and, when there is enough to fill a pot, both the sweat and the bullion are put into a "special gold Parke's pot" to effect a still further separation of the gold from the silver contained in the bullion. This reduces the cost of parting by obtaining the gold freer from silver.

Analysis of lead for gold and silver :—

After 1st Zn ..	.. Au 19 grs.
	Ag 70–80 ozs. per ton.
2nd 1st ..	.. Au trace, to ·5 grs.
	Ag 68–75 ozs.



ZINCKING PANS AND HOWARD STIRRER.

2nd Zn	Au none. Ag 20–30 ozs.
3rd Zn	Au none. Ag ·3–·4 ozs. per ton. Zn ·56 per cent. Cu ·00026 per cent. Sb ·00325 per cent.

The presses :—

1st gold press, bullion when retorted =	12ozs/1,000 doré Au bullion.
1st silver press	3,000 Ag per ton.
3rd zinc crust	150–200 ozs. Ag per ton.

When the "special Parke's pot" is heated, a "special gold dross" rises to the top, which is pressed off and sweated on a cupel, whence a copper slag is removed and sent to the smelters. Then 700 lbs. of spelter is stirred in, and a "special gold press" skimmed off. This is retorted for 12 hours, and a certain amount of zinc recovered. There are two products—"special gold retort bullion" and a dross. The former joins the "special gold dross" in the sweating cupel; the latter is first separately sweated, and then joins the other two. This rich bullion is then concentrated up to special doré bullion, which goes roughly 30–40 ozs. gold per 1,000 ozs. doré.

When the pan goes ·3–·4 ozs. silver it is clean, and the siphon (2 in. diam., ¼-inch thick) is put bodily in to get hot for 10–20 minutes. It is filled and placed in position, and the taps turned on. The lead runs by means of another channel iron gutter into the refining furnace—a reverberatory furnace in itself identical with the antimony and copper softeners. This furnace is braced, so as to resist very high temperatures, with buckstaves, each consisting of two lengths of railway iron. The refining operation is done to get rid of the zinc from the lead as a dry dross, which is skimmed like copper and antimony drosses. Here it is fired for five hours, so hard that the working doors get red hot; and, when cooled back, it is ready to skim. It is then fired until clean—the time from charge to discharge being 12 to 14 hours.

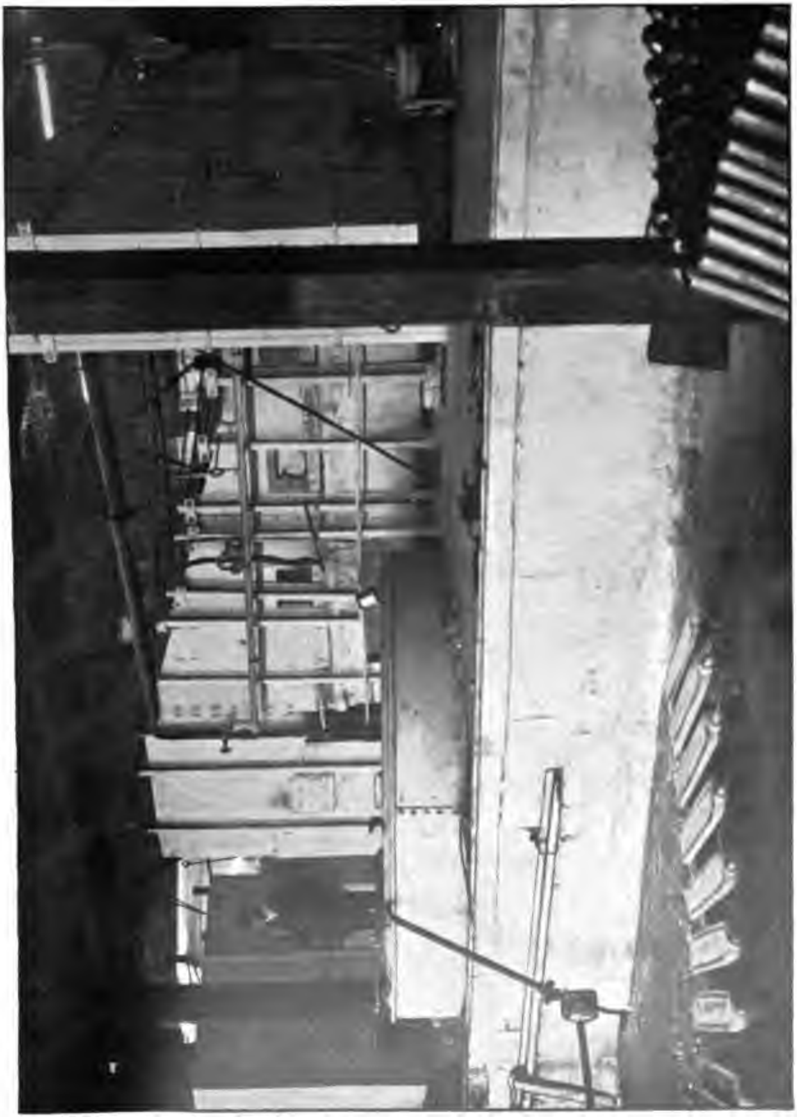
The lead is clean when a sample of well-skimmed lead in an iron mould shows no frosting on the surface. If there is much zinc in the lead from the Parke's pots the furnace is liable to crust up, owing to the formation of zinc accretions, which diminish the capacity, and require to be chipped occasionally. This is done soon after the charge is run down into the Merchant kettles, by lighting a fire in the hearth and hammering a crowbar into the crusts, which have formed mostly at the surface of the lead. Most of the lead is run into the Merchant kettle, where it is cooled to moulding temperature more economically than in the refining furnace. The small residue left behind becomes part of the next charge. The dross goes back to the smelters. The furnace requires the attention of one skilled labourer to fire and to skim.

Analysis of—

Zn Skimming.		Market Lead (average for 1 year).	
Pb	.. 85 per cent.	Ag	.. ·00096 per cent.
Zn	.. 10·9 per cent.	Cu	.. ·00026 per cent.
Ag	.. ·1–·2 ozs. per ton	Zn	.. ·00187 per cent.
		Sb	.. ·00325 per cent.
		Pb	.. 99·99366 per cent.

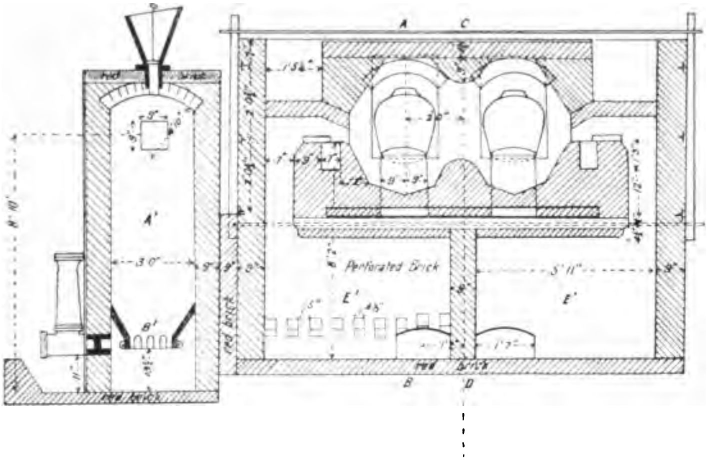
The Merchant kettles are reproductions of the zinc pans on a small scale, and are capable of holding 35 tons. The charge is fired on and skimmed of dross, which goes back to the refining furnace, is cooled back and run into moulds by means of a 1½-in. siphon tap of ¼-in. metal.

The stream of lead may be regulated by a nut and is run out through a sieve at the mouth of the tap, just hot enough to show no iridescent colours on the bars, into moulds, holding 100 or 200 lbs., placed in a ring about 10 feet from the tap. The 100-lb. bars are "English bars," and the 200-lb. are "China bars," which may contain ·5 ozs. of silver, which is paid for by the Celestials. One skilled labourer in the lead ring fills the moulds by manipulating an arm of the siphon tap, which is pivoted so as to turn about a spindle at the centre of the lead ring. With the other hand he skims the dross with a triangular skimmer. Another ordinary labourer

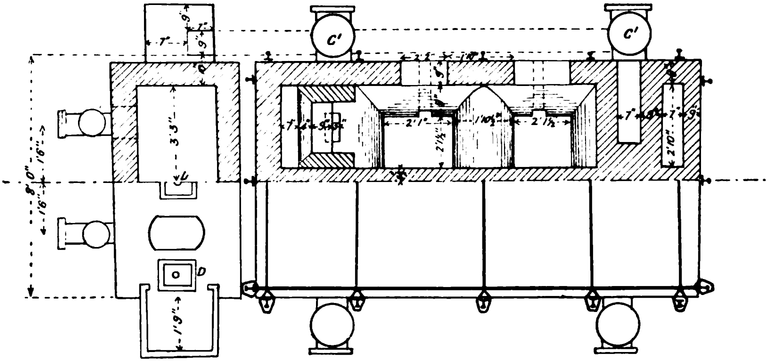


THE REFINING FURNACE

stamps the number of the charge, and stacks the lead ready for removal to the weighing machine and wharf. He also scrapes off all loose particles of lead from the bars. The time taken to mould a charge is about six hours.



— LONGITUDINAL SECTION. —

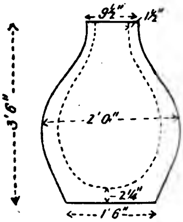


— PLAN & PLAN SECTION. —

FIG. 7.

SKETCHES SHEWING DETAILS OF RETORTS.

The retorts originally used were direct fired, but this method gave way to the more economic practice of gas-fired retorts, in which coal slack (an otherwise waste product) is made use of. A current of air is forced through the fine coal in A^1 by means of a steam jet introduced beneath the grate B^1, producing a water gas, which passes by means of one of two flues (8 in. diameter) into the retort area. The direction of draught is regulated by screw valves in the flues. These valves are changed every half-hour.



Dimensions of retort

Fig. 9.

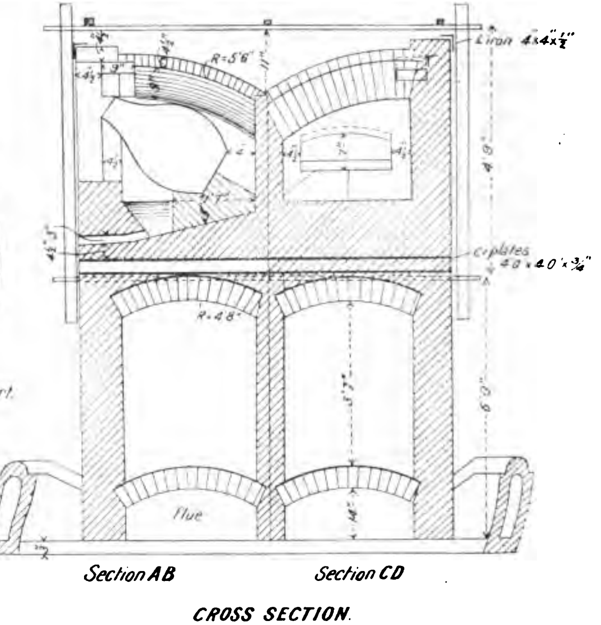
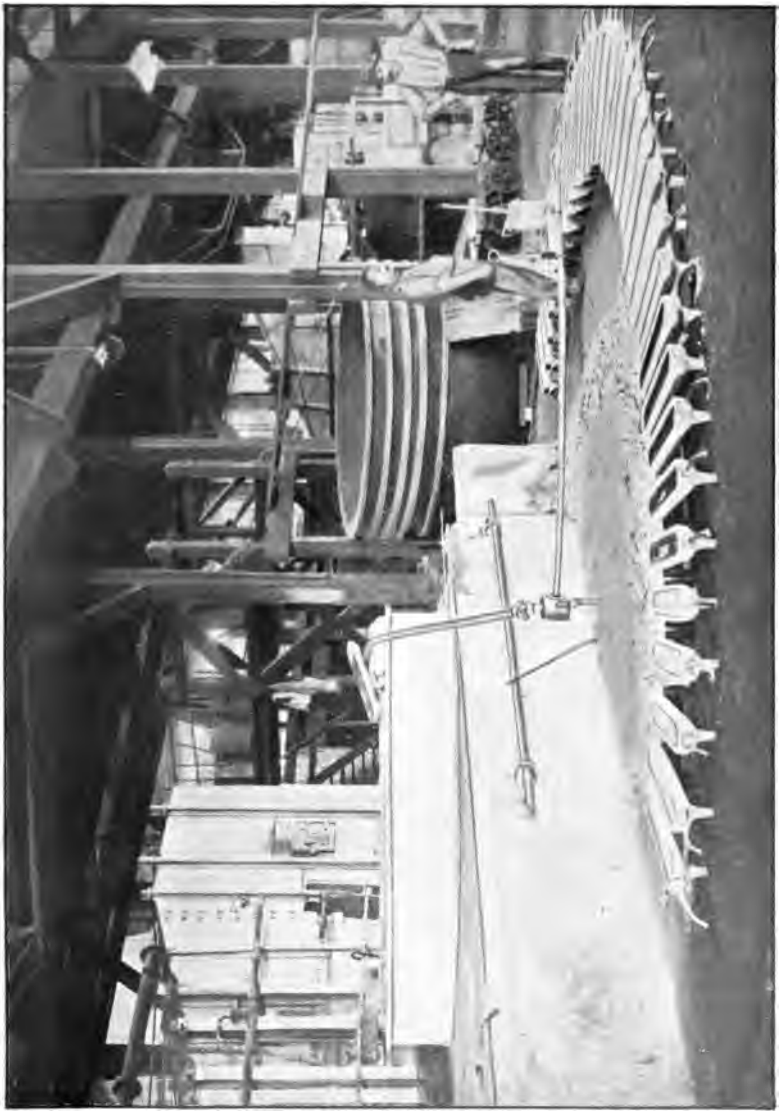


Fig. 8.
(Cross Section of Fig. 7.)

If the crust of coal forms a clinker, which is difficult for the steam to permeate, a long-pointed bar is put down through either of two "observation holes" to break the crust. The temperature of the grate is carefully regulated, the black mass only showing a glow here and there. This conditon gives the most economic use of the fuel. Air, previously heated by passing through regenerative flues

BAYLY
ON REFINING BASE BULLION.
PLATE VII.



THE LEAD RING,

filled with brickwork, E^1, is mixed with gas in the bottle area. The admission of air through a 15-in. iron pipe may be regulated to enter at the same part of the area as the gas by a butterfly valve.

The furnace itself consists of the producer and bottle area, made to hold two retorts. Two furnaces built back to back constitute a "nest." This gives economy of heat used.

The retorts are built in at an angle of 30° with the horizontal, and are supported at the base and neck by brickwork, designed to give the maximum exposed area to the hot gases consistent with stability.

Graphite retorts are used, made by Morgan, Battersea.

The condensers are cast-iron cylinders lined with composition, consisting of limestone and cement, to a depth of 2 in. These are supported by iron trestles on little four-wheeled trucks to enable them to be quickly and easily removed. There is a small outlet at the lower end for tapping condensed zinc.

Beneath the bottles, or retorts, there is an opening, to be used when the retort breaks, and the retort bullion runs in a cast-iron, brick-lined channel into moulds placed for the purpose, or on to the refinery floor. The bottles are now being used till they break. The final bullion charge is thus always caught. This ensures a greater length of use for the retort.

The retorts are first heated by lighting a small fire in the bottle area; the gas is slowly turned on, and when the bottles are hot the silver presses (previously broken up before removal to the retort furnace bin) are shovelled into the retorts with a special narrow shovel till full.

The bullion melts down, and more is added till the retort is full; the charge = 12 cwt. When all the charge is molten, a heated condenser is run into place and luted on with clay. The silver bullion is retorted until most of the zinc has sublimed (which takes about 12 hours); the condenser is then run back and the charge is skimmed. A trolley stacked with moulds is run into place, and the charge is dipped. The skimming and dipping take three-quarters of an hour, the whole silver retorting taking twelve hours (directly proportional to amount of zinc present).

There are three products—bullion, dross, and blue powder. About half a ton of bullion goes to the concentrating cupels, 2 cwt. of dross to the retort dross furnace bin, and the blue powder, which is the first zinc condensed in powder form, is taken back to the blast furnaces, to recover the silver it contains. Old retorts are chipped, and if the chippings are rich they are sweated on a bath of lead in the refinery; if not, they go back to the smelters. A certain percentage of zinc used is recovered and used again in the Parke's pots. The deficiency is filled up with German spelter when necessary. One skilled and one ordinary labourer are required for each nest of furnaces.

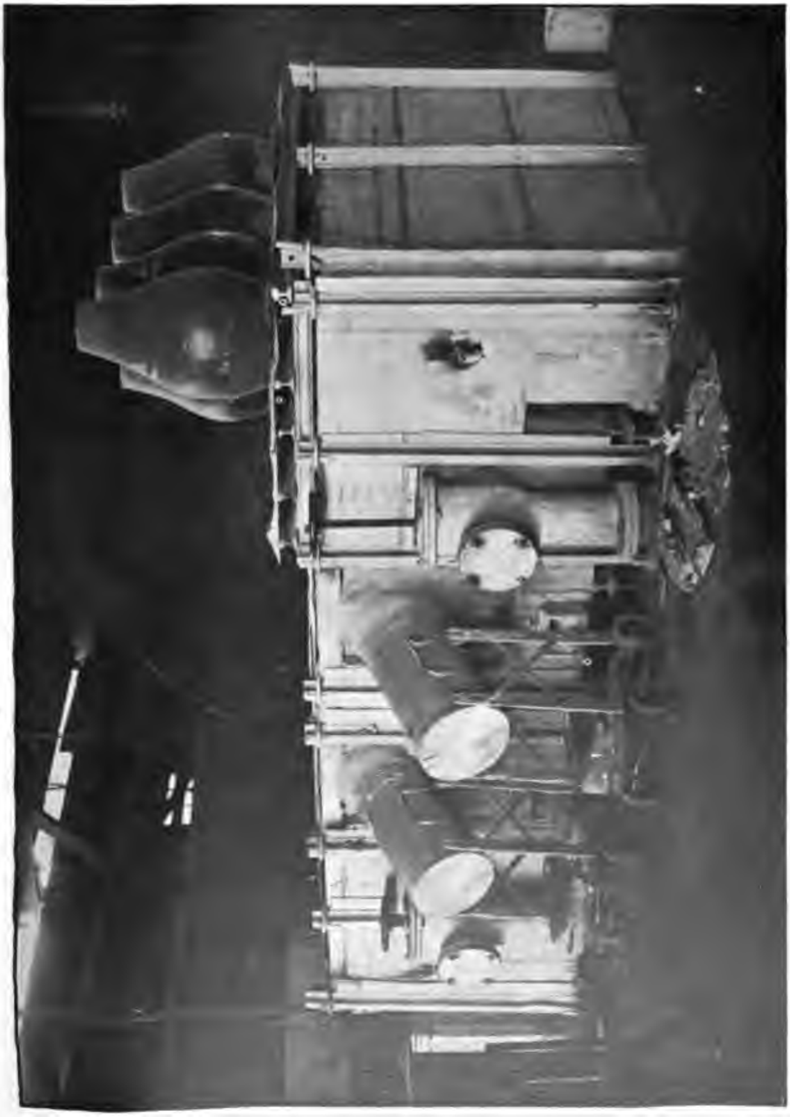
The bullion from the retorts is taken to the cupels for the concentration of the silver. The process consists of oxidizing the surface Pb to litharge, which overflows into a slag pot.

Analysis of—

Ordinary Gold Retort Bullion.	Dross (gold).
Ag 170–250 ozs. per ton	Ag 126 ozs.
Au 12 ozs. (1,000 ozs. doré)	Pb 60 per cent.
Silver Retort Bullion.	**Dross (silver).**
For Ag 3,200 ozs. per ton	Ag 2,400 ozs.
Au trace.	Pb 56 per cent.

Plate IX. is an illustration of the English cupel used at Port Pirie. The fire-box is situated on the left; the flames pass over a fire-bridge on to the surface of the metal, where they are met at right angles by a blast at 5 or 6 ozs. per square inch pressure, blown across to the door from the back of the cupel. The blast causes waves on the surface of the litharge, the level of the metal being just below the outlet channel, so that only the litharge is removed by the wave motion. For the escape of the gases down the flue there are three small openings to the right of the cupel. In front is the dipping door, and to the left of it the charging door, big enough to allow admission of a bullion bar. The test is replaceable, and when one is finished with a truck is run in beneath the supporting

BAYLY
ON REFINING BASE BULLION.
PLATE VIII.



THE RETORTS.

rails, and these are removed, and the test taken away. Relining a test consists in filling it with composition made up of—marble, 300 lbs.; crushed limestone, 120 lbs.; Portland cement, 120 lbs.; white fireclay, of putty-like consistency, 60 lbs.

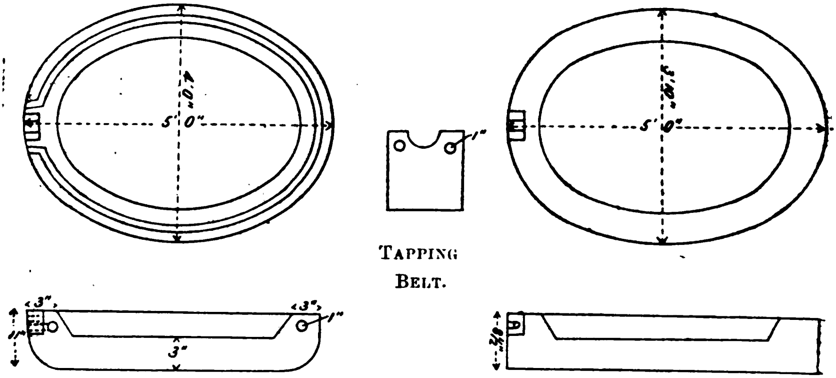
A wooden mould is placed on it and weighted down. The sides are pressed in with the fingers to free from air, and the cupel is covered with wet bags and put in a warm part of the refinery to dry. The new test is run in on a truck and wedged up into place, when the rails are replaced. The cupelling operation is carried on in three stages—in the concentrating cupel (concentrated to 16,000 ozs.), the refining cupel, and, lastly, the finishing cupel.

Analysis:—

Bullion from concentrating cupels ..	16,000 ozs. silver per ton of bullion.
Litharge from concentrating cupels ..	Pb 90 per cent. Ag 46 ozs. per ton.

The concentrating cupel has an inch water pipe running round at the litharge level to protect the lining. This is dispensed with in the other two, owing to the thick coating of pure silver which always forms upon it. All the cupels have, however, water-jacketed litharge taps. The brickwork of the furnace is supported by buck-staves and tie rods. The capacity of a test increases from 14 to 20 bars as the lining wears down.

The test is gradually heated up, and a charge of bullion put in, the fire made up, and the blast turned on. The time of operation varies greatly with the richness of the bullion. The amount of litharge run off is about five pots from the first charge, three and four from later charges; the reason for this is that the new test absorbs silver, and, therefore, the first operation takes much longer than succeeding ones to reach 16,000 ozs. The cupel litharge, above that required for the retort dross furnace, goes back to the smelters. Bars are continually added to keep the litharge at running level by the man who is required for the furnace. When the sample shows "half silver" the charge is baled with a red-hot baler into moulds, and taken to the refining cupels.



TAPPING BELT.

FIG. 10.

ELEVATION OF PLAN OF CONCENTRATING CUPEL.

REFINING CUPEL.

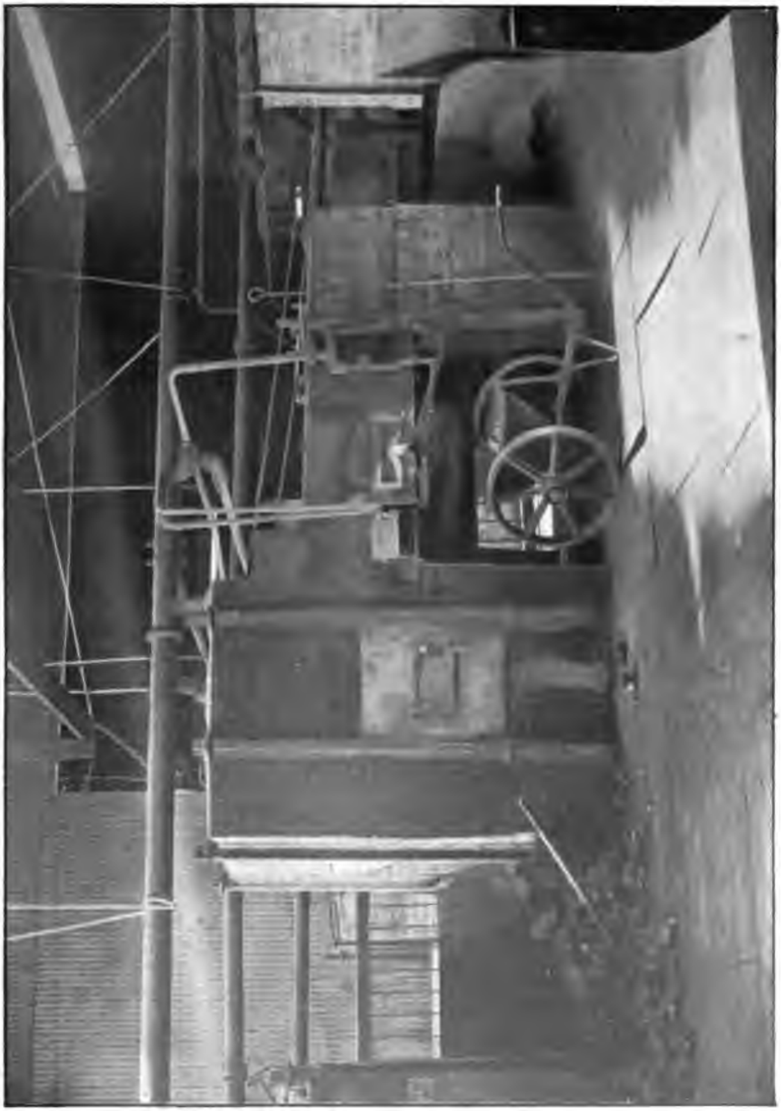
Analyses :—

Bullion from refining cupel ..	997·5 fine
Litharge	Pb 90 per cent Ag 55 ozs.
Bullion from finishing cupels ..	998·5–999·4 fine
Market Ag	997 fine

The life of the lining is variable, being on an average 45 charges. Experiments are continually being made with new composition, but that mentioned above, consisting of 70 per cent. limestone, 20 per cent. Portland cement, and 10 per cent. fire-clay, has so far proved the best. Old linings are treated exactly like old retorts, generally in the blast furnace. The refining cupel is shallower than the concentrator, although the shape and lining is the same. The time of operation, and also the amount of coal used, depends upon the condition of the test, which may hold from 14 to 24 bars of bullion.

When fired clean the surface of the molten silver has a transparent appearance, which is wanting when lead is present. Towards the dip the test is fired fiercely upon, and the ladle put in to get red hot. The charge is dipped into brick-shaped moulds, holding 1,000 ozs. roughly, weighed, and put into the finishing cupel, where it is "dried" on a new test, and any impurity left sinks into the lining,

BAYLY
ON REFINING BASE BULLION.
PLATE IX.



THE CONCENTRATING CUPEL.

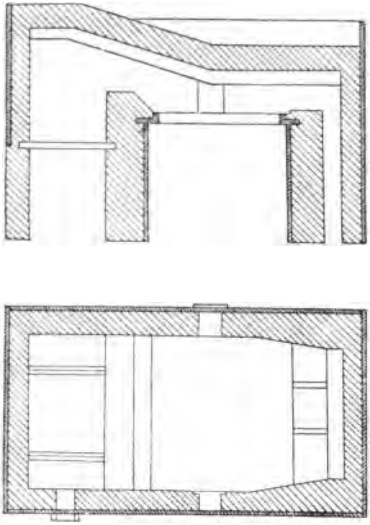


FIG. 11.

TYPE OF CUPEL FURNACE USED AT PORT PIRIE.

or is absorbed by powdered "compo," which is sprinkled on the surface. This is removed when sticky by adhering to an iron rod. The charge is dipped into octagonal moulds, three of which weigh about 2,050 ozs. This silver is too pure for the market, and so its fineness is reduced by adding granulated copper, to reduce it to "997 fine," and melting down in the graphite remoulding crucible—which is about 10 in. diameter and 18 in. deep, and just holds 2,050 ozs. easily. The fuel used is Australian coke, assisted by a low pressure blast. The crucibles are lifted from the furnace by an overhead bogey and pulley, and run on to an iron plate, whence the silver can be poured into a counterpoised mould on a balance.

Each mould must contain not less than 1,000 ozs., and not more than 1,050. While pouring, a piece of wood is kept burning in the crucible to prevent freezing. All loose pieces are filed off the bars, and, if over-weight, chisel marks are taken out. One end is hammered flat, and the number of the bar, the weight and fineness, and "B. H. P. Co. Ld." stamped on, and the bar is ready for shipment.

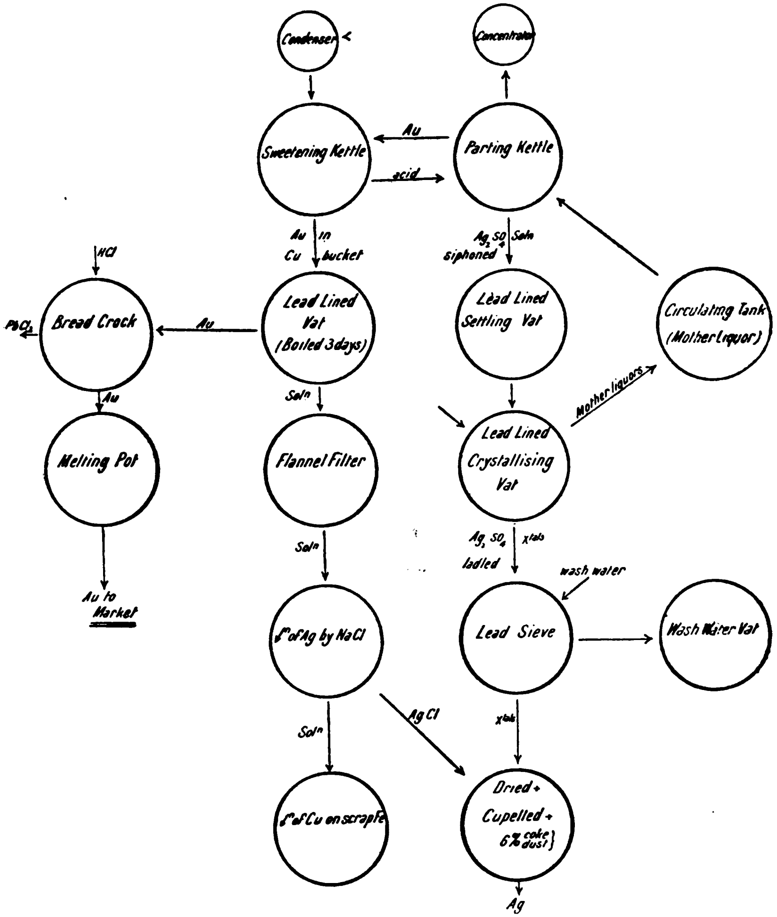


FIG. 12.

DIAGRAM OF GOLD PARTING PLANT.

The Gold Parting.—Every six months sufficient doré bullion has been stored to run the sulphuric acid parting plant economically. The acid is at present obtained from chemical works in South Australia in air-tight steel drums, at a strength 66° Beaumé.

Granulated bullion, or in the form of grids, was tried to obtain the best solvent action of the acid, but these forms produced too violent a frothing, and were displaced in favour of the shipping bars, which dissolved just as quickly and without effervescence. The kettle, one of three (of white cast iron, free from carbon, but containing phosphorus), is 3 ft. diam. and 3 ft. deep. It is lead lined, and fitted with a lead-lined overflow to the settlers in case of frothing. A lead-lined conical hood of cast iron, with an escape for volatilized acid to coke condenser and concentrator, is used. Into the two outside kettles the bars, to the weight of four or five thousand ounces doré, are charged in, and the kettle, three-parts filled with acid, is heated by a small fire. Here the bullion is allowed to digest for eight hours, then the fire is withdrawn and the gold settled. The acid, containing most of the silver as sulphate in solution, plus a little mechanically suspended gold, is siphoned through a 1-in. lead pipe into the "settling vats—lead-lined vats, 6 ft. x 3 ft. x 1 ft. 6 in.—where it is settled for 1 hour, passing thence by a water-cooled antimony tap into the crystallizing vats.

The tap (placed ½-in. from the bottom) is only in immediate contact with the acid during transference to the crystallizing vats, at other times being shut off by a lead strip in the settler, which covers the entrance. In the crystallizing vat the solution is brought to a density of 62° Beaumé by injecting live steam—this density giving the best precipitation. The vats are cooled by the circulation of water round a 3-in. jacket, placed all round the tank, and the silver sulphate crystallizes upon strips of sheet lead, 18 in. x 3 in., hung from wooden beams across the top, and is left to crystallize for 16 hours. The silver sulphate is scraped from these strips and ladled on to a leaden sieve of 1/16-in. mesh. The mother liquors are pumped up to the mother liquor circulating tank, whence they may be admitted to the dissolving kettles, and serve the purpose of cooling the acid from the centre or "sweetening kettle" ready for the next charge. The silver sulphate is washed with distilled water, dried, and cupelled with 6 per cent. of coke dust. The washings, containing 2½ per cent. silver sulphate, are run into the "wash water vat," 8 ft. x 8 ft. x 2 ft., steam heated, where the silver sulphate is precipated on copper tiles. The gold from the outside is transferred to the "sweetening kettle" and treated with fresh acid. After boiling some time, the sweet acid is siphoned into the side kettles to be further used.

Analysis for gold :—

Raw concentrates	..	..	About 6 grs.
Base bullion ..	..	..	1 dwt.
Doré bullion ..	..	..	40 ozs./1,000.

From the sweetening kettle the gold is removed by a copper shovel and bucket to a vat, where it is washed free of silver sulphate and ferrous sulphate, being in contact with diluted sulphuric acid and heated with steam for three days. At this stage the gold contains ·5 grs. silver per 1,000 ozs. The acid is drawn off and passed through flannel filters, which are subsequently burnt, to another vat where the remaining silver is deposited by a little common salt or zinc. This chloride is mixed with the silver sulphate on the cupel. Any copper in the solution, after precipitating the silver, is collected on scrap iron in another vat. The liquors from this vat are allowed to run to waste.

The gold still containing lead is put into a large bread crock, and treated with strong hydrochloric acid, and heated by steam. The solution, containing lead chloride, is drawn off and allowed to run to waste. The gold is dried, melted, and cast in 400–500-oz. ingots—a little sugar being put on the molten metal to give a bright face. The gold is of good quality, and will bring £4 4s. 7½d. per oz. on the market. The plant will only require the labour of two men.

Analysis of products of antimony dross furnace :—

Pb contains impurities—			Slag.		
Cu	..	·12	Pb	..	53·17 per cent.
As	..	trace	As	..	6·817 per cent.
Zinc	..	·0046			
Sb	..	·662	Sb	..	19·85 per cent.

The Treatment of the Antimony Dross.—Principle : partial reduction of litharge with carbon and absorption of antimony in the rest of lead oxide as antimoniate.

This is carried out in a reverberatory furnace of smaller dimensions than the copper softener, and differs in that it has no water-jacketed walls. Its capacity is 10 tons. It has three charging doors, but two tap-holes—one at the end at a higher level taps the slag first into a couple of cast-iron moulds, which catch any lead (the slag overflows into an enclosed space on the floor made tight with sand); the other at the side taps the lead.

A charge consists of 2,800 lbs. of broken-up skimmings, and requires 110 lbs. fine coal and 110 lbs. coke dust for reduction. It takes two hours to complete the operation—*i.e.*, four charges per shift. The slag is tapped once in 24 hours. The lead is tapped into a kettle and thence ladled into moulds, and is finally returned to the antimony softener. Two men are required on the furnace. The furnace lining just about lasts a campaign, which is three weeks or a month.

The antimony dross furnace slag is taken to the antimony blast furnace—Raschette type—with 10 tuyeres. To start the blast furnace, a wood fire is put in to warm the crucible and the water in the jackets is turned on. All the fire is raked out, and a 3-in. bridge of wood is placed over the crucible, and kindling wood put on to a height half-way up the jackets; 200 lbs. coke is added, then 15 bars antimony lead, 200 lbs. more of coke, and another 15 bars of antimony lead; then four charges of 100 lbs. coke, 450 lbs. antimony slag. The regular charge is then 100 lbs. coke, 450 lbs. antimony slag, 250 lbs. ordinary slag, 15 lbs. scrap iron, and so on till full. The fire is lit through the tuyere holes, and when caught a light blast is turned on, and, as the process goes on, the regular charge is run in. The ultimate blast pressure = 3–4½ ozs. The lead will be ready to tap two hours after charging, and the slag four hours after charging. A fresh charge is run in every quarter of an hour.

The lead obtained is "white metal," and finds a ready market, being used for type metal. The slag obtained goes to the smelters for retreatment. When the furnace is being run it requires the labour of five men.

Analysis of products from antimony blast furnace :—

White Metal.			Slag.		
Pb	..	76·04 per cent.	SbPb	..	6·21 per cent.
As	..	1·84 per cent.	SiO_2	..	30·5 per cent.
Zn	..	0·13 per cent.	FeO	..	26·92 per cent.
Cu	..	0·26 per cent.	MnO_2	..	3·80 per cent.
Sb	..	21·42 per cent.	CaO	..	9·50 per cent.
Ag	..	1 oz.	Al_2O_3	..	17·58 per cent.
			Zn	..	4·97 per cent.

Treatment of the Silver Retort Dross.—This is really a sweating operation on a large scale. The furnace is a reverberatory of about the same size as the antimony dross furnace. It is charged and skimmed like a copper softener. The lead is run into a kettle, whence it is ladled into moulds.

The bath consists of 200 bars of silver-clean lead (about 5 tons). When this is molten and beginning to oxidize, 750 lbs. dross is put in and roasted to free it of zinc. Then 750 lbs. from the cupels and 25 lbs. coal dust is added to reduce ; the remainder collects the impurities from the lead and silver. A fresh charge is added every two hours, consisting of—dross, 750 lbs. ; litharge, 500 lbs. ; coal, 25 lbs., till full, and then a skimming is taken once a shift. When skimmed clean the furnace is allowed to cool back, tapped into a kettle, and then baled into moulds. This bullion goes to the concentrating cupels, the slag to the smelters.

Analysis of products of retort dross furnace :—

Bullion ..	..	3,000 ozs. silver.
Slag ..	..	Pb 50 per cent Ag 300 ozs.

The Howard Press.—This consists of a cylinder through which pass two piston rods which rigidly connect this piston within to an iron basket below by two parallel rods, guided in grooves in the cylinder covers. The press is double-acting ; steam being admitted at the bottom forces the cylinder and ram into the basket; admitted on top, it raises the press. The whole thing may be raised by another steam cylinder and piston at the top. From this

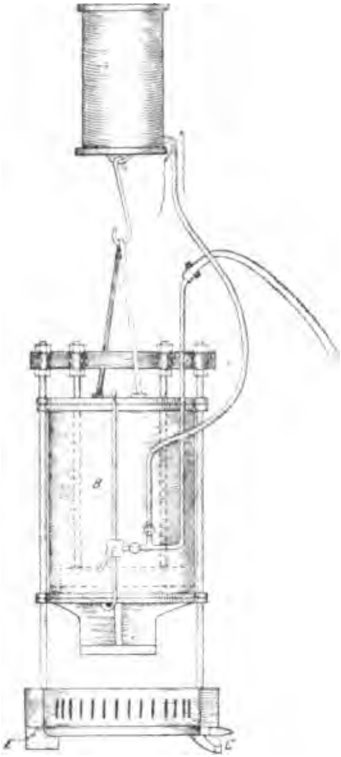


Fig. 13.
The Howard Press.

it is suspended to an overhead bogey running upon rails above the line of Parke's pots. Before the pan is cooled the press is raised (by means of the upper cylinder) and run over the pan. The steam in the upper cylinder now escaping lowers the cylinder B, and by admission of steam above the piston the basket is lowered into the pan to get hot, thus avoiding a thick coating of chilled lead. The pan is cooled back until a crust is formed, which is pushed by rakes over the basket. This is then raised just above the surface and is filled with skimmings with perforated ladles. The basket is then lifted by B, and the ram presses most of the lead out and compresses the skimmings. This operation proceeds again till the basket is full, when the press is lifted out and over the floor; the nut C is knocked loose, and the bottom of the basket, working on a pivot, E, is swung out of centre. The ram is forced down and the cake falls upon iron bars on the floor, is broken up, and transferred to the retort furnace bins. The area of the ram is about 12 in. diameter by 1½ in. thick. The dimensions of cylinder are—B, diameter 2 ft., height 3 ft.; of basket outside, diameter 2 ft. and inside 15 in., with slots about 3 in. x ½ in. The bottom is about 2 in. thick. The gold press is larger in basket proportions than this, as it has to be skimmed hot to effect a separation of the gold and silver, and, as the gold alloy crystals are finer, if too tightly pressed might force the gold back into the pan. This difficulty is not experienced with silver skims.

The Howard Stirrer.—This is another appliance lately introduced to overcome difficulties in stirring in the zinc. It consists of a hood

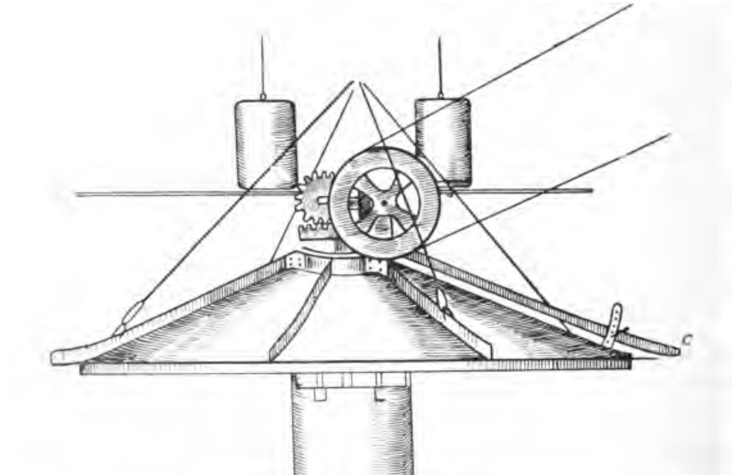


FIG. 14.
THE HOWARD STIRRER.

which closes in the pan; beneath this a paddle revolves about a vertical axis worked by bevel gearing and rope belting connected to a shaft. The paddle works in a small open-ended cylinder to cause connection currents and prevent the charge moving round as a whole, which would cause spilling. C is a lever whereby the stirring apparatus, irrespective of the hood, may be raised or lowered according as the pan is near the top (for gold zincs) or 3 or 4 in. down (for silver). These stirrers may be run along, like the presses, to any pan desired, and are counterpoised by lead weights. The stirrers are held down to the pans by extra lead bars. This is a great improvement on previous methods—hand stirring was very laborious, and steam stirring was liable to cause explosions unless the steam were absolutely dry. The modern method is just as efficient, requires much less manual labour and is not dangerous.

Vol. III. JUNE, 1907. No. II.

PROCEEDINGS—AUSTRALASIAN INSTITUTE OF MINING ENGINEERS.

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* THE MINING AND METALLURGY OF COPPER, SILVER, LEAD, AND ZINC,

AS SEEN DURING THE EXCURSION OF THE AUSTRALASIAN INSTITUTE OF MINING ENGINEERS, MAY, 1906.

BY FRANCIS W. SEWELL.

IN order that the processes seen may be followed in their proper order, the subject will be dealt with as follows :—(1) Wallaroo and Moonta Mines ; (2) Wallaroo and Moonta Mines Smelting Works ; (3) Broken Hill Mines ; (4) Concentration at Broken Hill ; (5) Smelting and refining of Lead, Silver, and Gold at Port Pirie.

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* Student's Prize Paper.

WALLAROO AND MOONTA.

The first acquaintance with the copper region was made at the Yelta shaft and dressing mill of the Parramatta Copper Mines Co. at Moonta. The country rock at Moonta is a hard, tough, felsitic porphyry, the copper lodes having a general strike of N. 30° E., dipping from 50° to 70°. This necessitates shaft-sinking on the lode, and underlay shafts are the rule on the field. The ore is raised in the Yelta shaft in wheeled skips, and consists of chalcopyrite, bornite, pseudomorphic hematite, containing, in addition, molybdenite and tourmaline. The ore from the bins at the brace is passed over a grizzly; the over-size is hand-picked into smelting ore, concentrating ore, and waste rock ("attle"), which is sent below for filling. The under-size is elevated to a trommel having ¾-in. holes. The over-size falls on to a picking belt made up of lengths of rope strapped with iron, working over wooden pulleys about 30 ft. apart; on each side sit boys who pick out smelting ore and "attle," and drop it into separate bins underneath. The middlings drop over the end, and are trucked with the fines from the trommel to the concentrating mill, which was not visited. The ore picked at the shaft and the concentrates from the mill are smelted in a blast furnace with limestone, in which a copper matte is produced. This matte is exported for further treatment.

The Wallaroo and Moonta Mining and Smelting Co.'s mines are situated at Moonta, and near Kadina. The Moonta division comprises several shafts, known as the North Yelta, Truers, and Moonta Mine. At Truers shaft specimens of carbonaceous rock were seen, which contained crystals of carnotite, found to be radioactive by Mr. S. Radcliffe, a metallurgist of the company. A concentrated sample, one-fifteeenth of the original ore, was found by Professor Bragg (Adelaide University) to have an activity 200 times that of $U_3 O_8$, an increase of three thousand. Further developments will be watched with interest.

The Wallaroo Mines.—The copper-bearing lodes in this division have a general trend east and west; the country rock being metamorphic mica schist (G 2.95), dark in colour, containing biotite, hornblende, and fluorite, the chief mineral of value being chalcopyrite associated with ironpyrite. The lodes being very wide in parts, and the country weak, it is necessary to keep the filling close up to the working faces, in which overhead stoping is adopted. Native timber, including mallee saplings, is used. At each level the ore is tipped into bins, and thence into skips through chutes. The mines are drained by two varieties of pump—a Cornish beam engine with a stroke of 6 to 10 ft. and 5 to 7 strokes per minute, and a set of electrically-driven Worthington pumps in Taylor's shafts at the 750, 1,230, and 1,830 ft. levels. The pumps are three throw, and raise about 250,000 gals. per day. The ore is trucked to grizzlies, and separated into fines and coarse; the latter going to the sorting plant, where it drops on to a picking belt, where smelting ore is hand-picked and sent to bins. The remainder falls into a gyratory crusher, and is broken to pass through a 2½-in. ring. A trommel then separates two classes—smalls, 1½-in. and under, and toppings, pass through 1½-in. square hole. The smalls go directly to the mill, while the toppings are re-crushed by rolls to pass through 1½-in. ring. The over-sizes are again hand-picked and separated into barren rock, smelting ore, and concentrating ore. The coarse ore, obtained from the first hand-picking, is re-crushed in a Gates breaker and trommelled; the coarse obtained is hand-picked to remove pyrite and low grade material, the remainder being sent to the smelters. The smalls from the grizzly are trommelled, the under-size (less than ½ in.) going to the Devon mill, the remainder to the sorting plant. The system of concentration at the Devon mill is very similar to that of Richman's mill at Moonta, which will be now described.

Concentration at Richman's Mill.—The ore from the storage bins falls into a Gates breaker, is crushed to 2 in. in size, automatically sampled, and passes into trommels with ¼ and ¾-in. holes, giving fines, which are sent to the mill, or to the smelters if rich enough, and coarse ore, which is hand-picked, separating "attle," smelting ore, and veinstuff, the last being sent to the mill. The mill comprises crushing and concentrating machinery, in which the ore is reduced by Blake crushers and then by Cornish rolls (Fig. 1).

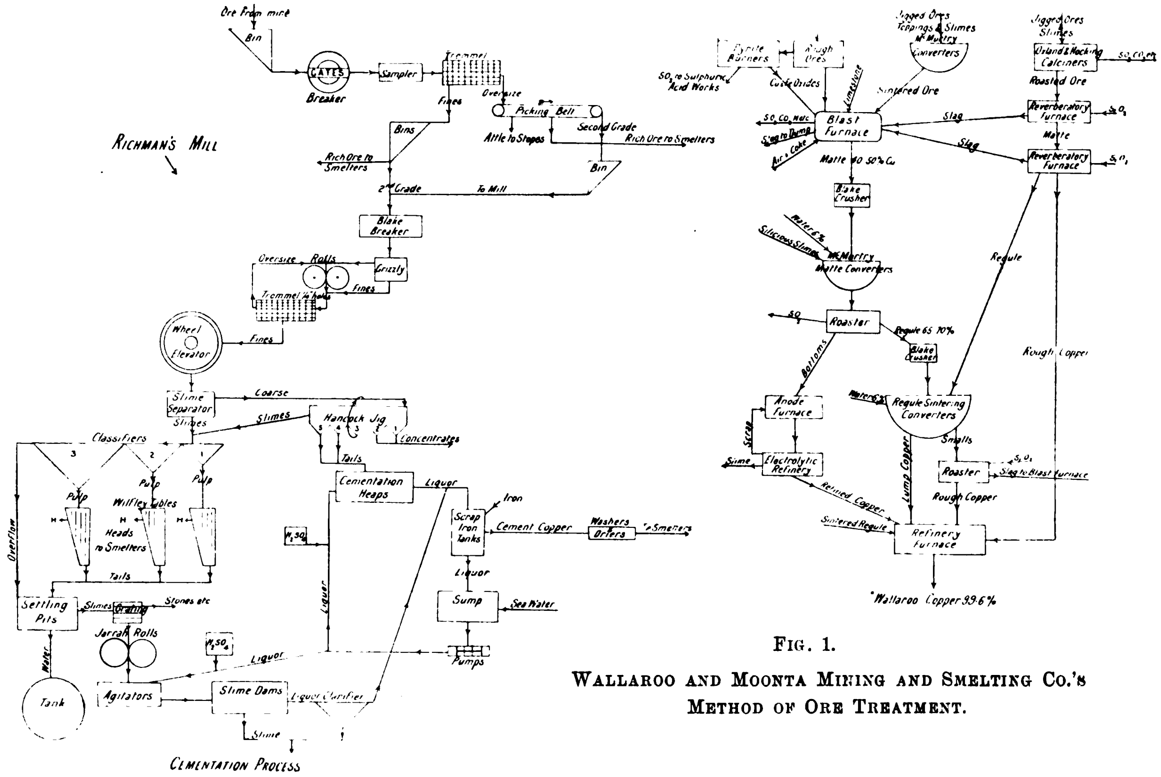

FIG. 1.

WALLAROO AND MOONTA MINING AND SMELTING CO.'S METHOD OF ORE TREATMENT.

Trommels with ¼-in. holes grade the crushed ore into two classes—over-size, returned to the rolls and under-size, which is raised by a large wheel elevator, provided with buckets on the inside of the rim, somewhat similar to an old-fashioned water wheel. The ore is delivered into slime separators, which are annular troughs, semi-circular in cross section, from which the slimes and water overflow through a central vertical shaft, which is hollow. The coarse material settles, and is forced up over the outside edge of the revolving trough by a baffle plate. The slimes are classified in a spitzkasten and the various products are treated on Wilfley tables; the heads obtained are fit for the smelters, the tails join the overflow from the spitzkasten and pass to settling pits. The coarse from the separators is fed into a Hancock jig, of which the screen, actuated by a system of levers, operated by cams underneath, moves upwards and forwards; down and backwards, causing a separation of heavy valuable minerals from the gangue; at the same time the screen bed rising and falling allows the former to pass through it into the hutches beneath. The screen is suspended in a long trough, the bottom of which is formed by five inverted pyramids or hutches. The first two from the feed end save rich smelter concentrates; the third a middle product, which is returned for re-jigging. The last two give tailings, which are sent to the cementation heaps. The overflow joins that from the spitzkasten. It is claimed for the Hancock jig that it has a classifying action which renders sizing trommels unnecessary, that it will handle unsized products ranging from ⅝-in. to 3 mm. on the same machine. Its capacity varies from 850 tons per day on coarse and 400 tons on fine material.

Cementation Process.—This solution method is used on the accumulated tailings and those at present produced (average, 9 per cent. of copper). To extract the copper acidulated sulphate of iron solution is the active agent. The tailings weather on exposure to the atmosphere and the copper becomes soluble, the action being accelerated by sulphate of iron and sulphuric acid produced in the heaps, also run on the top. The tailings are placed on prepared ground and surrounded by a drainage system which conveys the liquor from the heaps, which vary in height from 30 to 60 ft. and cover about 20 acres. The upper surface of the heaps is levelled and laid out into squares, 12 ft. x 12 ft., by low ridges, so that the solutions may be evenly distributed. The liquor applied was originally sea-water, to which sulphuric acid is added to give a ·2 to ·3 per cent. solution, which soon becomes charged with sulphate of iron up to 200 grains per gal., of which a large amount is deposited as hydrated oxide or basic sulphate in the interstices of the heaps. After percolation the liquor now containing copper sulphate and iron sulphate (about 20 grains per gal.) is collected by the drains and run by earthenware pipes to six tanks 60 ft. x 30 ft. x 3½ ft. deep, where it is clarified in the first two and passes over scrap iron in the others, thence through 3,000 ft. of wood-lined canals, 2 ft. 8 in. wide and 9 in. deep, filled with scrap iron, in which all the copper is removed.. Thus $Cu\,SO_4 + Fe = FeSO_4 + Cu$. The liquor flows to a reservoir and returns to the heaps by a set of acid-resistant Worthington pumps. The slimes produced at the mill cannot be leached, but after sufficient exposure in shallow dams are trucked to the agitation plant, where, on falling through gratings, which remove *débris*, &c., they pass into jarrah rolls, which break up the lumps, and thence into cylindrical wooden vats provided with revolving stirrers. Five tons of water from the precipitating canals (to which 18 lbs. of sulphuric acid per ton is added) is supplied to each ton of slimes. The pump is well stirred and run into settling pits, from which the clarified liquor containing any dissolved copper joins that from the heaps. The return pipes for the liquors are cast iron, wood lined, with rubber packing at the joints. The cement copper is removed from the tanks and canals periodically by diverting the stream and washing the iron. The copper is washed with clean water to remove iron, etc., dried, and sent to the smelters.

The Wallaroo Smelting Works (See Fig. 1).—The products delivered to these works include rough ore from the sorting plants, dressed ores (as jig concentrates), toppings, slimes, cement, copper, and purchased ore, which are sampled on arrival. The first operation, which differs with the nature of the ore, is the desulphurization. Some high grade "roughs" go direct to the blast furnace; the remainder are fed into pyrite burners, so as to utilize the sulphur in making sulphuric acid. The sulphurous gases are freed of dust and pass to the acid works where 100 to 120 tons of chamber acid (G 1·62) are made per week. The burners are arranged in three double rows with 6 furnaces back to back in each, fitted with square fire-bars capable of rotating. The burnt oxide is sent to the blast furnace. The fine ores are either treated in M'Murtry converters or Oxland-Hocking revolving calciners; the product of the latter is fed into a reverberatory reducer, producing a matte which is roasted and melted to rough copper, and a rich regule which goes to the sintering converter. It is intended to dispense with the calciners and instal more converters. M'Murtry converters are segmental cast-iron pots, 6 ft. to 9 ft. in diameter, supported on horizontal trunnions, capable of being tipped. A perforated plate at the bottom covers the entrance for compressed air, which is supplied through a flexible hose. The converter is first warmed by a small coal fire, and the charge tipped in, consisting of 30 per cent. toppings, 30 per cent. slimes, 30 per cent. jigged concentrates—the sulphur contents being 18 to 20 per cent.; 5 to 10 per cent. water is added to a 9-ton charge. The air blast is gradually raised till it reaches 12 to 16 ozs. In 13 hours the desulphurization is completed, and the slaggy, porous mass is tipped, cooled with water, and broken up. The product goes to a water-jacketed blast furnace which produces a copper matte (40 to 50 per cent. Cu), limestone being used for fluxing. The slag is granulated by a jet of water and runs to the dump. The matte is crushed in a Blake crusher to ⅜-in. and goes to the matte converters; the charge being—matte 66 per cent., silicious slimes 30 per cent., water 6 per cent. The charge weighs 8 tons, and is blown for 8 to 12 hours, during which a large amount of sulphur dioxide escapes up a hood to a flue provided. The sintered coherent mass is broken up and charged into a reverberatory furnace, where it is melted and roasted until some copper is formed, which will carry any gold, silver, and impurities down with it. The copper bottoms, on being tapped, are melted down in the anode furnace into plates suitable for the electrolytic refinery, where the copper is purified by electrolysis. The slimes or impurities fall to the bottom and are removed; the refined copper is re-melted in a refinery furnace.

The roasted regule, containing 65 to 70 per cent. Cu, is crushed to ⅜ in., moistened with 6 per cent. water, and charged into the M'Murtry conical converters, holding 2 tons; air blast of 5 ozs. pressure is used for 8 hours, when the spongy mass of copper and partially reduced residue is tipped and broken up. The lump copper goes to the refinery furnace, the rest to a roaster, where rough copper is obtained; the slag is sent to the blast furnace. Care is taken to prevent fusion, thus blocking the perforated plate. The lump and rough copper is melted down with sintered regule in the refinery furnaces, and cast into ingots and plates. The output for 1905 was 6,500 tons of copper, 99·6 per cent. The sulphuric acid made is mainly used in converting phosphate rock into "superphosphate," a mixture of $Ca\,SO_4$ and $Ca\,H_4\,(PO_4)_2$ at works close by.

Broken Hill and Port Pirie.

Mining.—The mining at Broken Hill is confined to a low ridge two miles long, about 200 ft. above the desert, formed of schistose slate and garnetiferous sandstone. The outcrop was manganese ironstone, 20 to 100 ft. wide, traversing the centre of the ridge. The oxidized ores continued to a depth of 300 to 500 ft., when the sulphide zone was reached. Various methods of mining were tried at first, but finally the square-set system was adopted. In this was used oregon, from Oregon and British Columbia, sawn 10 in. square. The sets were built up from wall to wall square in three directions to the stopes overhead. As the depth increased and the lodes maintained their width, "creeps" or earth movements crushed the timbers, causing fires and loss of life and money. Filling up the existing sets with mullock was resorted to, and by packing it well up to the working face safety was insured. The "block" system of removing rooms 9 ft. by 9 ft. and then filling up with waste has been successfully adopted. The Broken Hill

Proprietary Co. worked out a large amount of ore to a depth of 300 ft. by means of an open cut three-quarters of a mile long, 360 ft. wide, and 300 ft. deep, which also provides an enormous amount of rock for filling underground, besides relieving the pressure from above. In the oxidized zone, portions of which are still worked by some companies, cerussite (with embolite), cerargyrite, iodyrite, kaolin very rich in silver, pyromorphite, &c., formed the main wealth, while the sulphide ore is an intimate mixture of galena, rhodonite, blende, garnet, and quartz, occurring with garnetiferous sandstone, calcite, carbonates of iron and manganese, fluorspar, and green feldspar. To the north of Broken Hill lie the Copper Blow and Silver Peak claims, which, judging from a personal inspection, promise to be heard of in the future. Outbreaks of fire in the Northern group of mines necessitated their flooding, as the most efficient means of extinction. This was done from the Water Co.'s mains. Many million gallons of water were poured down, and left there for a few weeks to do their work. It is now being raised to the surface as rapidly as possible, and stored in reservoirs. The Broken Hill Proprietary Co. isolated a portion of their mine, affected by fire, with bulkheads, through a pipe in which the escaping gases pass behind a brattice or wall of canvas plastered with clay, dividing the drive, which leads the gases to an upcast shaft provided with a fan. Horses, about 60 per shift, are used in this mine in the main drives, &c., for pulling trucks. The main drive runs in the country parallel to the lode; crosscuts connect with drives in the lode. Passes, &c., are built up as the stopes advance, and are surrounded with filling kept up as close to the face as possible.

Metallurgy.—The general method of wet concentration adopted on the field includes breaking by Gates or Blake crushers, Cornish rolls, ball-mills, or Heberli grinders; sizing by trommels, jigging, classification by spitzkasten and treatment on vanners. The products from the above consist of galena more or less pure, zinciferous tailings, and slimes. These are further treated by flotation, granulation, magnetic and sintering processes. A detailed description of one mill visited will serve to illustrate the principles of wet concentration generally adopted. Reference to the accompanying flowsheets will show the modifications in various mills.

The Broken Hill Proprietary Co.'s Mill—(Fig. 2).—The ore is raised to the surface in trucks holding one ton, which run on a raised tram line to the mill, attached to an endless rope, bringing the full trucks and returning the empty ones. The trucks are tipped by a tumbler—a steel frame provided with two circular bands on rollers. The ore falls on a grizzly, 12 ft. long, inclined at 50°, with a one-inch space between the bars. The fines fall into

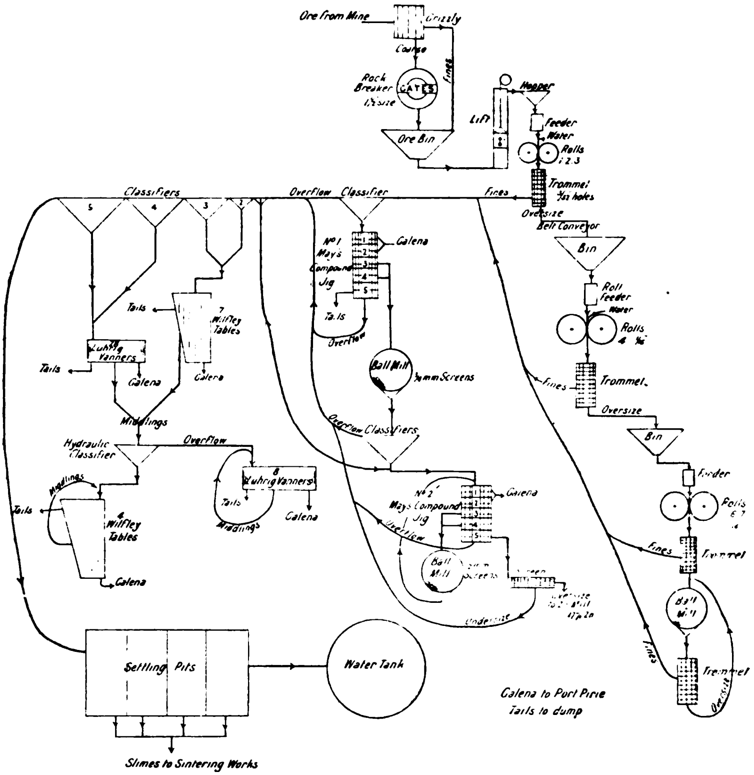

FIG. 2.

BROKEN HILL PROPRIETARY CO'S CONCENTRATING MILL.

a bin; the coarse passes through four No. 4 Gates breakers, is reduced to 1½-in. size, and then it falls into the same bins as the fines. The mill treats 12,000 to 13,000 tons per week. The ore is then raised by a lift and fed into Cornish rolls by a chute provided with a roller. This slowly revolves and moves the ore evenly forward. A jet of water mingles with the ore above the rolls, which are 2 ft. 6 in. in diameter, 15 in. broad, running at 37 revolutions per minute, at ⅝-in. apart. The rolls are held in place by thick rubber discs, one roll having flanges, between which the other fits. The pulverized material enters a conical trommel, with $^3/_{32}$-in. holes. The under-size flows to the classifiers. The over-size is elevated, screened to remove some fines, and fed into the No. 4 rolls (45 revolutions per minute) at $^5/_{16}$-in. apart. Trommelling gives fines, which pass to the classifiers, and over-size, which passes to the Nos. 5, 6, and 7 rolls (80 revolutions per minute) at $^1/_{16}$-in. apart. The product is again trommelled. The over-size is partly returned to the same rolls, and partly to rolls running in contact, or to a Krupp ball-mill, crushing to $^3/_{32}$-in., running at 30 revolutions per minute. The product is again trommelled, the over-size is returned, and the fines passed to the classifiers. The hydraulic classifiers remove the slimes from the coarser material, the former going to a series of spitzkasten. The coarse passes into a No. 1 compound May jig, which has two rows of five compartments, terminating below in a point and spigot. A series of plunger compartments 15 in. wide extend down each side of a central partition. The remainder of the top of the compartments is closed by woven wire screens 6 in. below water level, suitably supported. The plungers are 3 ft. 6 in. by 14 in., provided with clacks 2 ft. 6 in. by 6 in., which open on the upward stroke of the plunger, but close on the downward, giving a quick lift to the water in the screen box and a slow return. The plungers are worked by levers from a shaft which is moved by a lever and eccentric, giving 165 ¾-in. strokes per minute. The concentrate in the first and second hutch is sent to the smelter, that from Nos. 3 and 4 is re-ground in the ball-mill; the tailings pass to the dump. The ball-mill has ¾ mm. screens, and the product is classified; the slimes flow into the spitzkasten, the coarse being rejigged on fine jigs running at a higher speed. This gives a concentrate for the smelters; a middling product is re-ground to ½ mm. in a ball-mill, and returned to the same jig. The tails are screened, the fines going to the spitzkasten; the over-size (Zn. 17 per cent.) is sent to the zinc mill. The spitzkasten are in groups of five, and settle fine material from the true slimes, which overflow to the settling pits, whence clear water is obtained for the vanners, &c. The settled material in No. 1 spitzkasten is treated on the fine jigs; in Nos. 2 and 3 it goes to seven Wilfley tables, run at 216 strokes per minute, where smelter concentrate is produced. Tails go to the dump, and a middling, which is re-classified and treated on four Wilfley tables, produces smelter concentrates and tails, the middling being returned to the head of the machines. The overflow from the classifier is treated on eight Luhrig vanners, the belt being of balata, and run at 150 strokes per minute, treating six tons per day. These give fine smelter concentrate and tails. Nos. 4 and 5 spitzkasten give fine sand, which is treated on twenty-eight Luhrig vanners, producing smelter concentrate, tails, and middlings, which join that from the first Wilfleys. The products of the above machines are:—Coarse jigs—first hutch, lead 70 per cent.; second hutch, lead 56 per cent. Fine jigs—lead 40 per cent. Wilfley and Luhrig vanners—lead 50 per cent. The average assay of all products is lead 55 per cent.

Sintering Works.—The slimes are excavated and trucked as a pasty mud to the sintering works, about 6 miles from the mine, on the Silverton line.

Donald Clark ("Australian Mining and Metallurgy") gives the following as a general analysis of the slimes:—

Galena	$Pb\,S$ 24 per cent.	
Blende	$Zn\,S$ 29·4 per cent.	
Pyrite	$Fe\,S_2$ 3·38 per cent.	
Ferric oxide	$Fe_2\,O_3$ 4·17 per cent.	
Ferrous oxide	$Fe\,O$ 1·03 per cent.	in rhodonite and garnet
Manganese oxide	$Mn\,O$ 6·66 per cent.	

Alumina	Al_2O_3 5·4 per cent., in garnet and kaolin
Lime	$Ca\,O$ 3·4 per cent.
Siliea	$Si\,O_2$ 22·9 per cent.
Silver	·06 per cent.

The following analysis was kindly supplied by Mr. Bradford :—

	Before Sintering.	After Sintering.
Lead	18·83 per cent.	16·5 per cent.
Zinc	16·6 per cent.	12·0 per cent.
Sulphur	13·0 per cent.	7·3 per cent.
Silver, per ton	18·83 ozs.	17·4 ozs.

Five per cent. in weight is lost in sintering. The slimes are sun-dried, in layers 6 in. thick, and then cut into rough bricks 10 in. square, with shovels, and separated. When sufficiently consolidated they are stacked in heaps 220 ft. long, 20 ft. wide, and 7 ft. 6 in. high. At the bottom, at intervals of 10 ft., a few sticks of wood are placed in small flues connected with a central flue running nearly to the end of the heap. The bricks are carefully handled, being friable; about 10 per cent. are broken. The fines are separated and worked up with water in a pug mill, and plastered over the heap to exclude the air. The stack is burnt by lighting the firewood on the windward side and then on the leeward side. The stack first sweats, sulphur distils out, and volatile salts condense on the outside. When the pyrite has ignited the air supply is regulated by closing the flues, no more fuel being added—about 1 per cent. being the amount used. In two or three weeks the heap will have burnt and cooled down, when it is broken out, and the dark grey and black sintered product trucked to Port Pirie. Any unburnt and fine stuff is added to the next heap. Losses occur by volatilization, &c., and amount to zinc 5 per cent., lead 17 per cent., and silver 12 per cent. of that originally present. Mr. Carmichael (B. H. P. Co.) suggests that manganese from the rhodonite plays an important part in the oxidation, also that calcite is converted into calcium sulphate, and produces lead sulphate, which oxidizes some galena. Donald Clark ("Australian Mining and Metallurgy") considers these changes doubtful, and proposes that the slow oxidation of the sulphides produces sulphur trioxide, which attacks the unaltered sulphides, and argues that the interaction of sulphides and sulphates would result in the desulphurization of the slimes. The methods of concentration adopted by other mills differ mainly in arrangement and the fine grinders and vanners used. At the North Broken Hill Proprietary mine (Fig. 3) Hancock jigs are used, also Heberli disc grinders. (See Proceedings Aust. Inst. M. E., April, 1906, "Concentration of Silver-Lead Ores," by V. F. Stanley Low, where the system adopted at the "Block 10" is described).

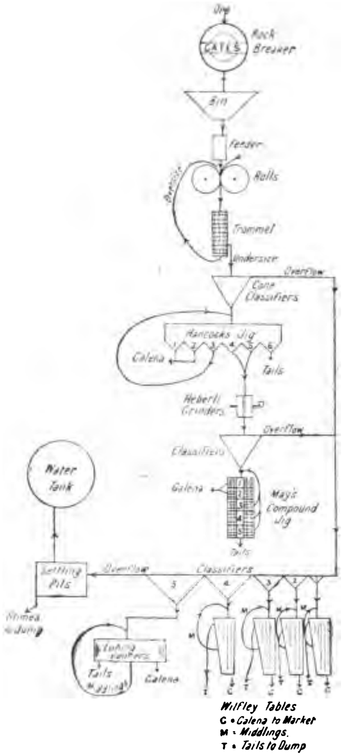

FIG. 3.

NORTH BROKEN HILL CONCENTRATING MILL.

The vanners used are Wilfley, Card, Krupp, and Phœnix-Weir, all similar in action; also, Warren and Luhrig belt vanners.

The tailings produced by the mills average—lead, 3 to 6 per cent.; zinc, 9 to 30 per cent. (usually 16 to 20 per cent.); silver, 2 to 9 ozs. per ton. The gangue is rhodonite, quartz, garnet, sandstone, calcite, &c., and various processes are in use to separate the blende from the gangue. The accumulated tailings amount to over 4,000,000 tons.

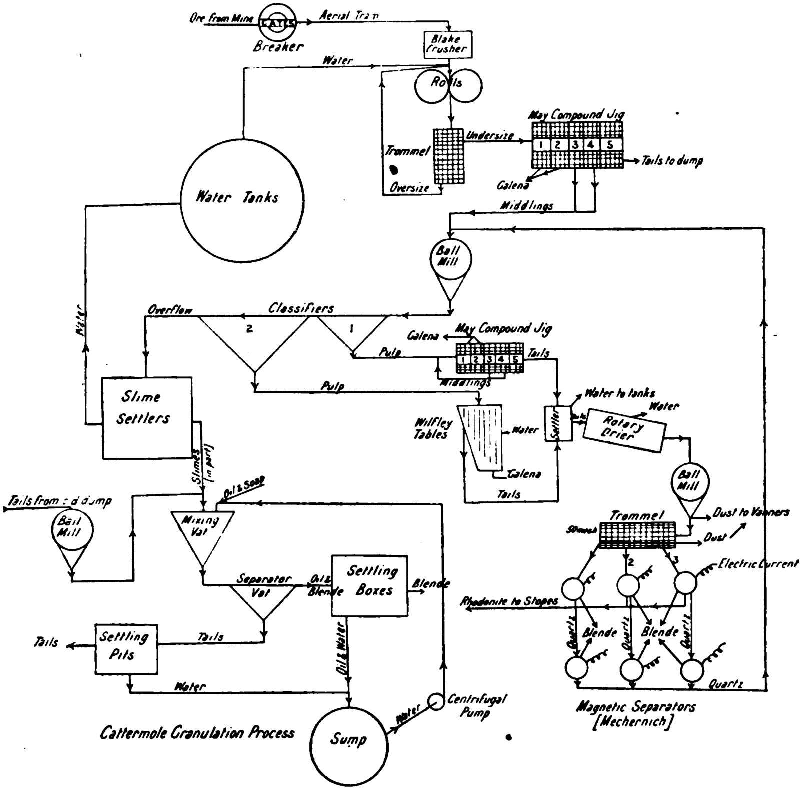

FIG. 4.

BROKEN HILL CENTRAL CONCENTRATING MILL.

MAGNETIC AND GRANULATION PROCESS.

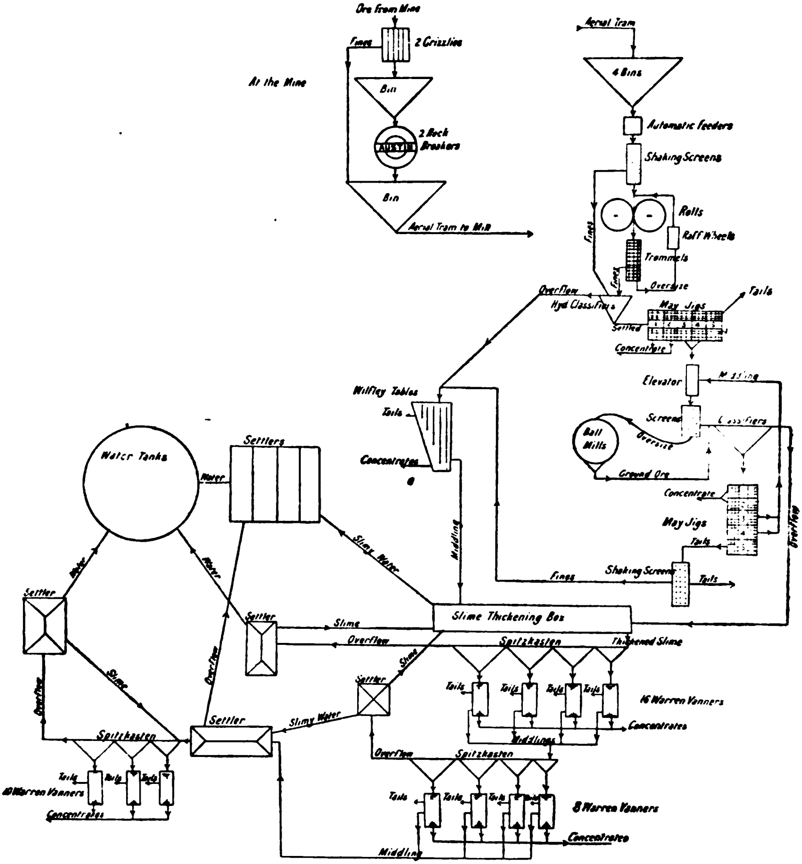

FIG. 5.

BROKEN HILL PROPRIETARY BLOCK 10 CONCENTRATING MILL.

The Mechernich Magnetic Process at the Central mine (Fig. 4) deals with tails from the fine jigs and vanners working on an ore originally assaying silver 12·6 ozs. per ton, lead 17·9 per cent., and zinc 9·1 per cent., in a gangue of garnet, sandstone, and rhodonite. After being dried on a conical drier, the tails are crushed to 1 mm. in a ball-mill; the product is trommelled into three grades—over 1 mm., ½ mm. and above, and less than ½ mm. Dust is removed by fans, and treated wet on vanners for a lead concentrate. The material passes into the magnetic machines, which consist essentially of two parallel magnetic poles, one cylindrical, built up of thin layers of zinc and iron internally wound, which revolves above a fixed pole. A brass chute pressing gently against the revolving pole feeds the ore evenly between the poles; the para-magnetic minerals—rhodonite, garnet, and blende—feebly cling to the revolving pole, while the galena and quartz (dia-magnetic) fall straight down into a chute. The former group of minerals pass to a field of decreasing intensity on the revolution of the pole, and the blende first falls off into a chute; the garnet and rhodonite finally fall off into their receptacle and are clean enough to be sent below for filling. The blende is shipped away. The quartz-galena product passes to the lower floor, and is re-treated on another set of machines to extract a little remaining blende, then it is returned to the concentrating mill to remove the galena. From 21,000 tons of middlings, assaying zinc 27 per cent., lead 8 per cent., and silver 9 ozs. per ton, concentrates were obtained assaying zinc 39 per cent., lead 12 per cent., silver 13·4 ozs. per ton, at a cost of 6s. 9d. per ton. The Central mine (Sulphide Corporation Ltd.) is also treating accumulated slimes and tailings by the Cattermole granulation (see Fig. 4) process. This process is described in the U.S.A. Pat. 763,260 as "a process of separating metalliferous matter from gangue, consisting of agitating a mixture of powdered ore with oil in emulsion with water containing an alkaline emulsifying agent so as to agglomerate the oil-coated particles into granules, and subjecting the mixture to classification to remove the small non-coated particles from the granules." At present oil, soap, and sulphuric acid are used, making the process one more like flotation than granulation. Any coarse material is ground in a ball-mill, which delivers the product into the boot of a bucket elevator, into which the slimes are also tipped. The ore, averaging 20 per cent. zinc, is raised to the top of the building and agitated with oil, soap, and acid in a dilute pulp in a conical vat provided with horizontal stirrers, the temperature of the solution being at about blood heat. The mixture is then run into spitzkasten, the delivery pipe projecting below the surface of the water; the tails drop to the bottom and fall through a spigot into settlers, whence they are sent below for filling. The overflow from the spitzkasten, consisting of oil-covered particles of blende and galena, runs into settlers. The blende is shipped away. The water from the settlers is re-used. Most of the oil adheres to the blende and is lost.

Flotation Processes (Fig. 6).—The flotation process of C. V. Potter for recovering the zinc is used by the Zinc Corporation on tailings purchased from the British Broken Hill Co. This process depends on the fact that when the tailings are mixed with a dilute acid solution at 65° C., the sulphides present become coated with gas bubbles, and, their specific gravity being lowered, rise to the top and may be removed, the gangue being unaffected. Several theories have been advanced, which will be briefly mentioned. Their correctness is a point still to be settled. One theory suggests that carbon dioxide from the calcite attaches itself to the sulphides and raises them. A. K. Huntington (paper read before the Faraday Society, 12th December, 1905) assumes that the effective carbon dioxide in the flotation of Broken Hill tailings is derived from the iron and manganese carbonates which are decomposed at 65° C. by dilute acid. J. Swinburne and G. Rudorf, in a paper read before the same society, suggest that the selective flotation is due to the presence of "greasy" sulphides as galena, and that it is a problem of surface tension and adhesion. De Bavay's theory is that fluorspar present is decomposed by the acid, giving hydrofluoric acid; this attacks silica present, forming silicon fluoride, which, in contact with water, decomposes into silicic acid and hydrofluosilicic acid.

$$\mathrm{Ca\,F_2 + H_2SO_4 = Ca\,SO_4 + 2\,HF.}$$
$$\mathrm{Si\,O_2 + 4\,HF = Si\,F_4 + 2\,H_2o.}$$
$$\mathrm{3\,Si\,F_4 + 4\,H_2\,O = H_4\,Si\,O_4 + 2\,H_2Si\,F_6.}$$

Silicic acid is gelatinous, and forms granules with the blende, and his is raised by the gases present to the surface. Delprat's process, milar in effect to Potter's, uses a dilute solution of sulphuric acid nd salt-cake (sodium bisulphate). The mechanical appliances for

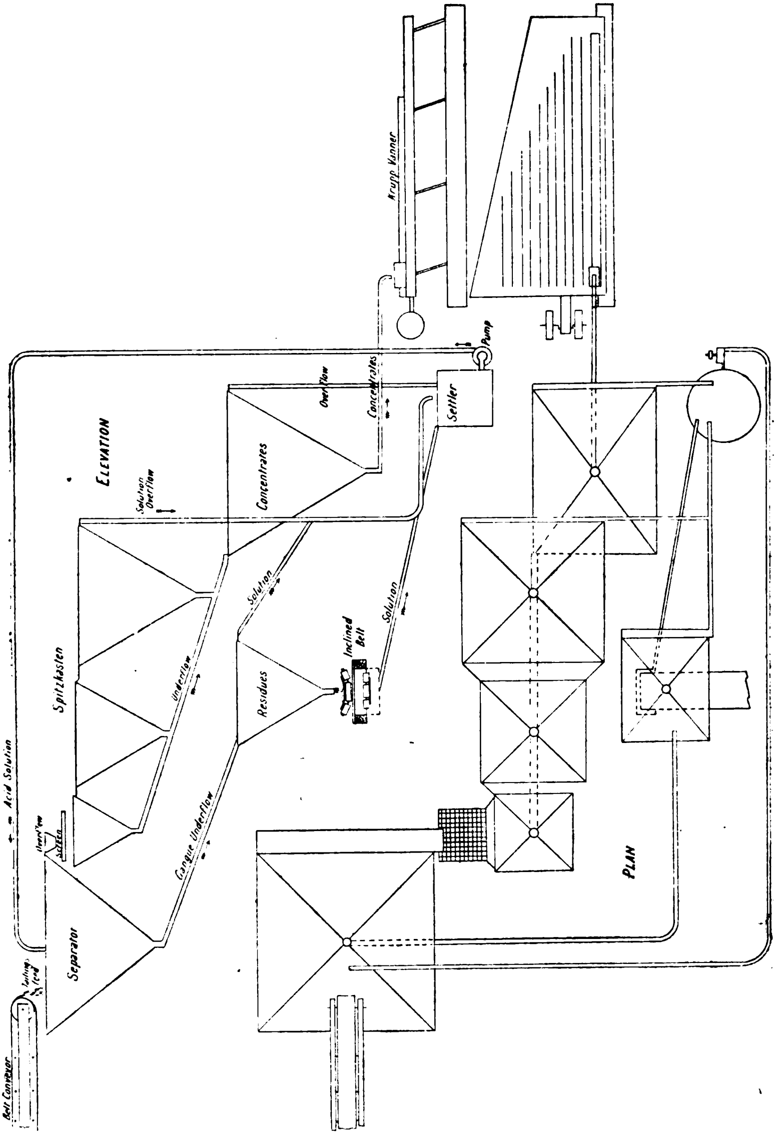

Fig. 6.

Zinc Corporation Concentrating Mill—Potter's Process.

effecting the separation are practically the same, with a slight modification in the Delprat process, in which a double-pointed spitzkasten is used for the flotation instead of a single-pointed one used by the Zinc Corporation. The tailings are fed into a wooden vat or spitzkasten by a travelling belt in a steady stream. The vat is filled with acid solution (sulphuric acid 2·5 to 3 per cent.) heated by steam to 65° C. or over. A separation immediately takes place; a black scum of blende with a little galena rises to the surface and is carried off by the overflowing stream. The gangue falls to the bottom and discharges into a launder. The floated particles flow along a launder through a screen, to remove *débris*, thence over a succession of "drops" to disengage the gas bubbles, into spitzkasten, in which the blende settles, the clear liquor running into a sump. The spitzkasten discharges on a Robins belt conveyer set at an incline; this drains away the adhering liquor and drops the blende into bins. The tails from the separator vat are settled and fall on to an inclined belt conveyer, which drains and discharges them at a suitable spot for sending underground. The Zinc Corporation runs the concentrate over a Krupp table and recovers a little galena and removes some sand entangled with the blende. The concentrate averages zinc 37 to 46 per cent., and lead 3 to 10 per cent., while the residues assay zinc 3 to 4 per cent., lead 2 to 4 per cent.

The Broken Hill Proprietary Co.'s Smelting Works at Port Pirie (Fig. 7).—The products received include coarse and fine galena concentrate, the first grade assaying Pb 60 per cent., Zn 10 per cent., S 16 per cent., Ag 30 ozs. per ton; second grade, Pb 45 per cent., Zn 12·5 per cent., S 14·5 per cent., Ag 22 ozs. per ton; and sintered slimes, Pb 21 per cent., Zn 17 per cent., S 13 per cent., Ag 13 ozs. per ton. The fluxes consist of dense metamorphosed limestone, shipped from Turton Point, and pure hematite from the company's quarry at Iron Knob. Coke is shipped from Newcastle. The coarse concentrate and slimes go to the blast furnace direct, but the fine concentrates require a preliminary treatment to agglomerate and desulphurize them. This is effected in the Huntington-Heberlein (H-H) process, in which the fine ore is mixed

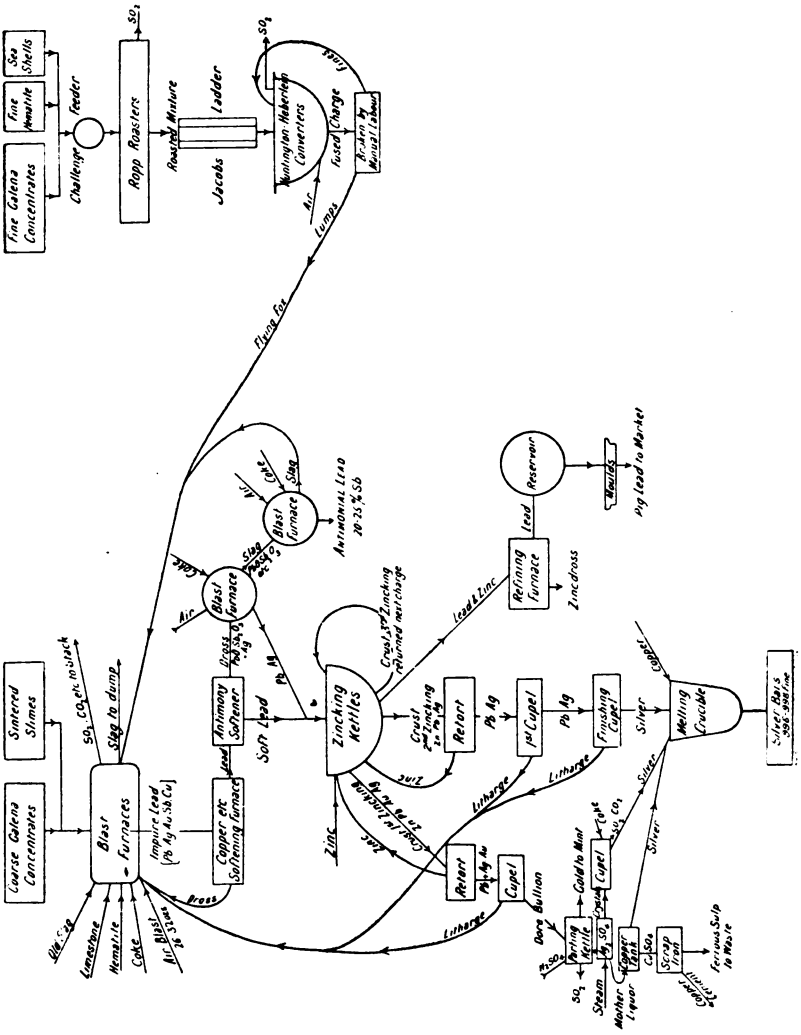

FIG. 7.

BROKEN HILL PROPRIETARY CO.'S SMELTING WORKS, PORT PIRIE, S.A.

with fine sea-shells from the adjacent coast and fine hematite in proportions of—sulphides 15 cwt., silicious ore 160 lbs., shells 240 lbs., hematite 100 lbs. The sulphides average lead 55 per cent., zinc 10 per cent. The mixture is roasted in Ropp mechanically-rabbled reverberatories, and the product discharged contains 8 per cent. sulphur. Each furnace treats 80 tons per day. The rabbling is done by rabbles supported by a horizontal bar connected through a narrow slit in the mid-line of the hearth to a travelling carriage below, worked by an endless rope. The rabbles are returned to the feed end of the furnace outside, and so cooled down. The furnaces are heated by four fires at one side, the heat being directed by special construction well into the furnace. The roasted ore is elevated, while hot, in trucks above the H-H converters, which are segmental cast-iron pots 8 ft. in diameter and 6 ft. deep, holding from 5 to 8 tons, and swinging on horizontal trunnions. Two feet from the bottom of the pot rests a perforated plate which covers the vent leading to the compressed air pipe. Over the plate is a short frustum of a cone, perforated, covered by another punched plate. A conical hood, with a telescopic pipe 2 ft. in diameter, connects with a main flue leading to the main stack, and can be lowered over the pot, taking away the sulphurous gases (S 8 per cent.) evolved. The converter, being hot from the previous charge, is filled with the hot granular material, and the blast gradually turned on till it reaches 24 ozs. Sulphuric and sulphurous anhydrides are given off freely, and the mass glows brightly. In about 4 hours the operation is finished, no more sulphur being smelt coming off. The mass is porous, care being taken to prevent complete fusion. The slaggy mass is a basic silicate of lime and lead, the other metals being oxides. About 2 per cent. sulphur remains, which passes into the blast furnace slag. When cool, the charge is tipped on to iron cones below and broken up with hammers; the lumps are raised by a "flying fox" (aërial tram) to the smelters; the small amount of fines produced is returned to the converters. There is practically no loss of lead or silver due to volatilization, and very little to the escape of zinc. The limestone is converted in the roasters almost entirely to sulphate. A large amount of lead also forms sulphate. The re-actions are subject to discussion, those given by the patentees being—

(1) At 700° C. $CaO + O = CaO_2$.
(2) At 500° C. $4 CaO_2 + PbS = PBSO_4 + 4 CaO$ (on cooling down).
Thus 1 and 2—$4 CaO + PbS + 4 O = PbSO_4 + 4 CaO$.
(3) $3 PbSO_4 + PbS = 4 PbO + 4 SO_2$ (melting point).

Donald Clark (*E. and M. Journal*, 20th October, 1904), also C. Guilleman (in the same journal, 10th March, 1906), consider these equations unsatisfactory, owing to calcium dioxide being easily decomposed by heat, and suggest that Ca O and Pb S (intimately mixed) when heated will react thus—

$CaO + PbS = CaS + PbO$.
$CaS + 2 O_2 = CaSO_4$.
$CaSO_4 + PBO = CaPbO_3$ (calcium plumbate) $+ SO_2$.

the result being oxidation and desulphurization. The blende is probably desulphurized thus—

$ZnS + 3 ZnSO_4$ (formed in roasting) $= 4 ZnO + 4 SO_2$.

Professor Borchers (*E. and M. Journal*, 21st September, 1905) also favours the theory of the formation of calcium plumbate, but this is disputed by Maynard Hutchings (*E. and M. Journal*, 21st October, 1905), who considers that the sulphuric anhydride is the active agent, calcium sulphate acting as a carrier of oxygen. The Bradford-Carmichael process (described *E. and M. Journal*, 28th October, 1905) dispenses with the preliminary roast and uses dehydrated gypsum direct. It was used by the Broken Hill Proprietary Co., but the plant is at present lying idle. Another similar process is that of Savelsberg (*E. and M. Journal*, 9th December, 1905). The blast furnaces are the water-jacketed, cold-top type, treating 230 tons of ore each daily. The blast is supplied by Green's improved blasters (Root's type), at 26 to 32-oz. pressure. The water for the jackets is pumped from the sea by a centrifugal pump supplying 5,000,000 gallons daily. The flues are provided with dust chambers and connect with an iron stack 205 ft. high and 41 ft. in diameter at the base. A typical charge is as follows:—

H-H mixture	1,100 lbs.
Hematite	1,100 lbs.

Sintered slime	800 lbs.
Slag (non-zincky, from old dump)	800 lbs.
Limestone	500 lbs.
Raw concentrates	400 lbs.
Coke	280 lbs.

No matte forms, and the slag is tapped every twenty minutes into slag bowls and tipped over the dump. The lead collects in the crucible and automatically discharges through a "siphon" or pipe connecting the bottom of the crucible to a bowl outside, whence it is tapped into moulds holding 75 to 80 lbs. of bullion. The slag runs high in zinc. Samples are taken out of each bar of lead with a small pick—one out of the top and one out of the side—and mixed into lots representing 50 tons of lead. Each blast furnace will treat 2,000 tons a day of ore, fluxes, and fuel, the slag averaging :—

$\mathrm{Si\ O_2}$	25 to 27 per cent.
$\mathrm{Fe\ O}$	31 per cent.
$\mathrm{Mn\ O}$	5 to 5·5 per cent.
$\mathrm{Ca\ O}$	15·5 to 17 per cent.
$\mathrm{Zn\ O}$	13 per cent.
$\mathrm{Al_2\ O_3}$	6.5 per cent.
S	3 to 4 per cent.
Pb (wet assay)	1·2 to 1·5 per cent.
Ag	7 ozs. per ton.

The lead is trucked to the refinery, where the gold, silver, antimony, a trace of copper, and other impurities are removed, and soft marketable lead produced. It is fed into the first softener ; a reverberatory furnace with a deep hearth, large fireplace, and short length, holding 1,500 bars of lead. The lead melts, and the heat is raised until oxidation commences ; the copper-lead alloy rises to the surface, is oxidised with a certain amount of lead. Other entangled impurities rise to the top and oxidise. The fire is then checked, and the litharge, becoming pasty, is raked off through the working doors. The lead is then tapped into the antimony softener, and the dross is returned to the blast furnace, where the copper passes into the slag. The antimony softener is worked similarly to the first softener; the antimony oxidises, and forms lead antimoniate, which dissolves the litharge formed. The dross is smelted down at intervals in a small blast furnace, with a little powdered coke, which reduces some lead carrying the silver contents of the dross. This lead goes to the zincking kettles. The slag is run down with coke in the same blast furnace at intervals, producing antimonial lead containing 20 per cent. Sb.

De-silverizing Process.—The lead is tapped into large segmental cast-iron kettles, heated underneath, and zinc is added—about 1 lb. for every ounce of precious metals present—and stirred for 15 minutes by revolving stirrers. These stirrers can be moved from kettle to kettle by a travelling crane. The kettle is covered by an iron hood during stirring, which keeps away the air and prevents splashing. The stirrers are then removed and the lead cooled. The zinc rises to the top alloyed with practically all the gold present and some silver and lead, and when semi-fluid is removed by a Howard skimmer. This is a short cast-iron cylinder with a perforated hinged plate closing the bottom. A piston can be forced into this cylinder by compressed air, which squeezes out the semi-fluid lead from the semi-solid zinc. The skimmer is hung over the kettle by a chain from a crane, and is dipped under the surface of the metal and the crust pushed into it until full. The cylinder is raised out of the lead, and its contents squeezed by the piston. The crust is then forced out on to the floor beside by opening the hinged bottom. The crust is all removed and the lead re-heated and re-zincked with crusts from the third zincking of a previous charge. This operation removes practically all the silver, but is repeated to remove any remaining traces; the crusts obtained in the last operation being again used as above indicated. The lead, containing ·75 per cent. Zn, is siphoned into the refining furnaces (reverberatory), where the zinc and some lead are oxidised by hot air, and removed. The soft lead is tapped into a cast-iron reservoir and thence siphoned into moulds holding 1 or 2 cwt. The zinc crusts, containing 2,500 to 3,000 ozs. of silver per ton, and 12 to 14 per cent. of zinc, the remainder being lead, are fed into bottle-shaped retorts holding 12 cwt., in a regenerative furnace heated by producer gas, generated from coal in a producer alongside. The air supply is heated by the regenerator, the gas being already hot. This ensures a saving of 25 per cent in the use of solid fuel. The gold and silver crusts are kept separate. The gold crusts are allowed to accumulate until in sufficient quantity, the gold contents of the ore being small. In ten hours the zinc has distilled over and condensed in a receiver in front. The residue of lead and silver (or gold) is ladled into moulds and taken to the cupellation furnaces, which are short reverberatories. The hearth consists of an elliptical test mounted on a movable truck. The test is composed of crushed limestone, clay, and sand, well mixed and rammed into an iron frame. Corrosion of the upper edge is prevented by water circulating through a copper pipe embedded in the test. Air under pressure is directed on to the surface of the molten lead, and assists oxidation, driving the litharge towards an outlet at the opposite end, whence it flows into conical moulds mounted on wheels. The cupellation is done in two stages. In the first the test is kept filled until a very rich alloy is produced. This is then finished off in a similar furnace inside an enclosure. The silver remaining is melted into crucibles and cast into bars 1,000 to 1,050 ozs. in weight. As the silver is extremely pure, a little copper is usually added to make an even fineness of 996 to 998. The crusts from the first zincking are retorted and cupelled, and yield Doré bullion (1 of gold to 25 of silver). This is parted with sulphuric acid in cast-iron kettles heated underneath, and the silver dissolving forms sulphate of silver, sulphur dioxide being carried off by a hood—

$$2\,Ag + 2\,H_2\,SO_4 = Ag_2\,SO_4 + H_2O + SO_2.$$

The gold remains, and the acid is siphoned off into another vat, where it is diluted with steam. Large yellow crystals of silver sulphate are precipitated, and are put on filters and well washed. The liquors are then run over copper and the remaining silver extracted; the copper passing into solution is recovered with scrap iron, giving cement copper.

$$Ag_2\ SO_4 + Cu = Cu\ SO_4 + Ag_2.$$
$$Cu\ SO_4 + Fe = Fe\ SO_4 + Cu.$$

The sulphate of silver is mixed with charcoal, and reduced and melted in a cupellation furnace—

$$Ag_2\ SO_4 + C = Ag_2 + SO_4 + Co_2.$$

The gold is well washed and smelted. Parting is done twice a year. The ores and furnace products are assayed for lead both by the crucible and ammonium molybdate methods. Silver and gold are estimated both by crucible and scorification assays. The purity of the silver produced is estimated by titrating a nitric acid solution of the silver with potassium sulpho-cyanide solution, using iron alum as indicator. Potassium ferro-cyanide is used in the estimation for zinc.

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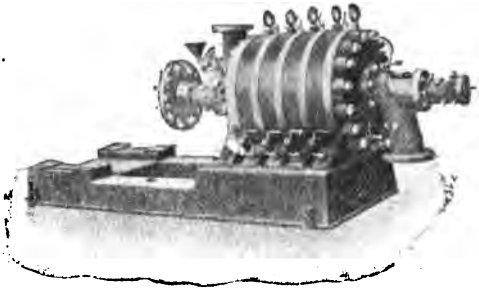

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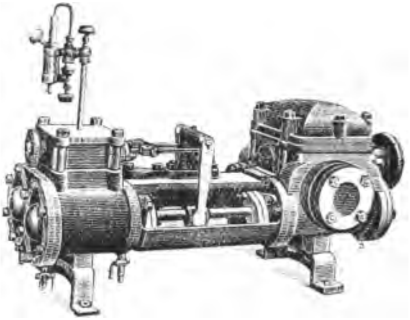

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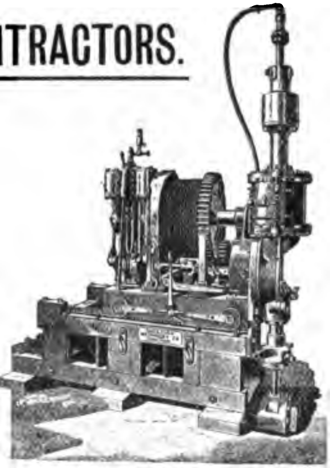

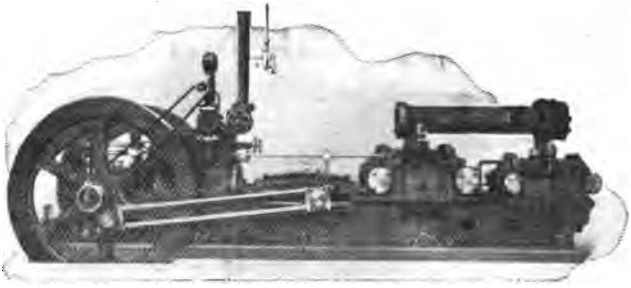

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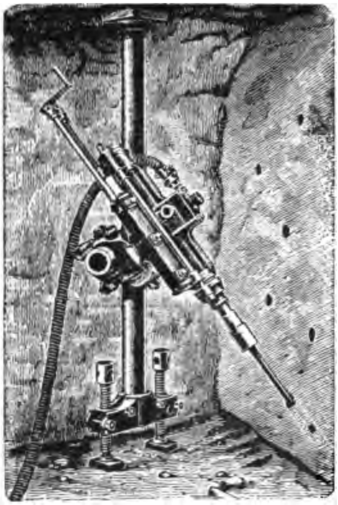

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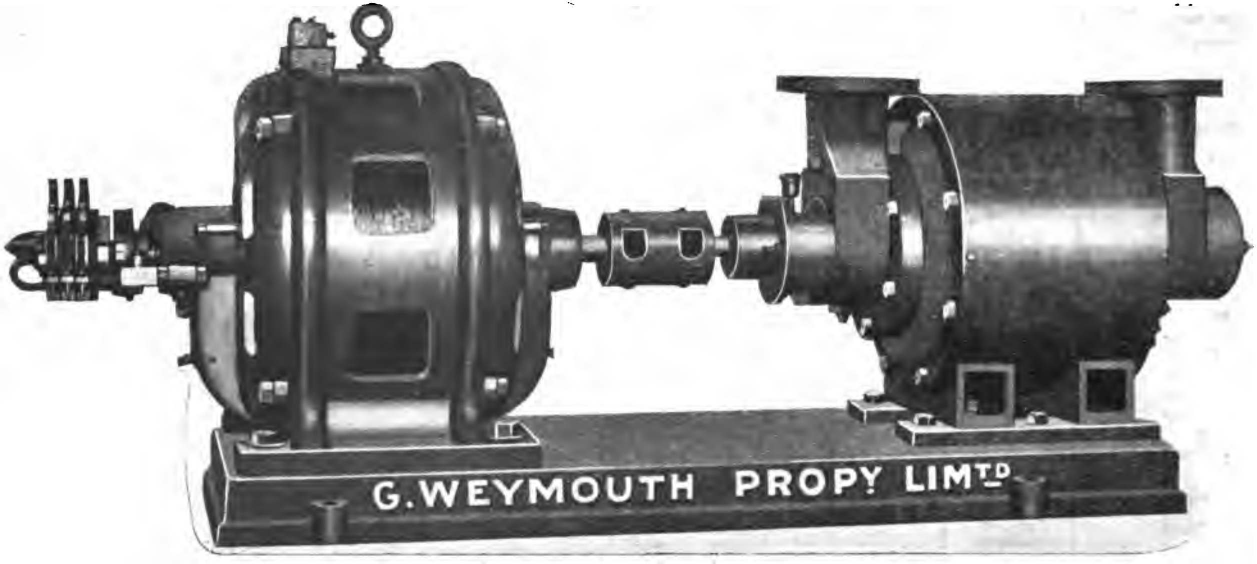

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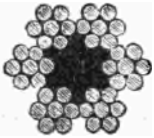

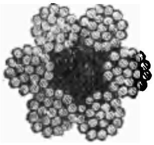

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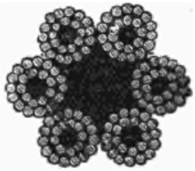

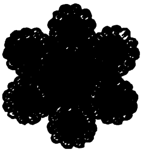

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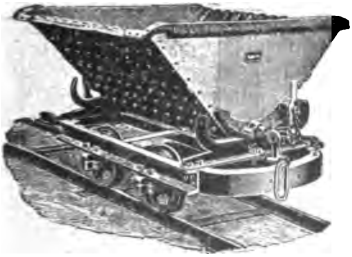

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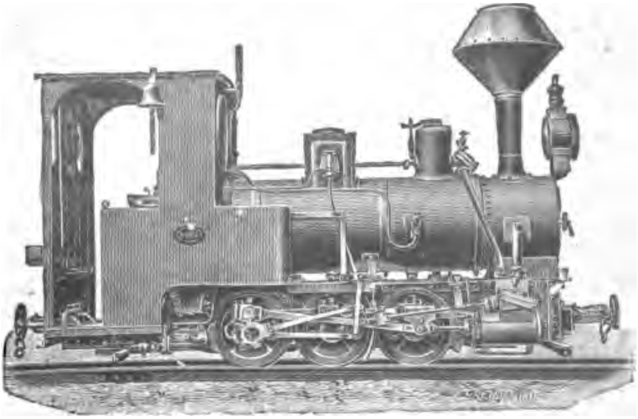

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